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CONTENTS

	PAGE	RUSS. PAGE
We Shall Fulfill the Tasks of the Seven-Year Plan Ahead of Time	91	1
THE BLAST-FURNACE INDUSTRY		
Improved Preparation of Small Coke. V. V. Vinogradov	93	3
The Automatic Control of the Composition and Temperature of Gas over the Horizontal Cross Section of the Blast Furnace. Z. I. Nekrasov and Ya. M. Obodan	95	5
Charging Equipment with Double-Wall Cone. L. L. Kordabnev	97	7
Reducing the Sulfur Content in Pig Iron. P. M. Matveev	100	10
THE STEELMAKING INDUSTRY		
Experience on the Application of Open-Hearth Sinter. B. G. Petukhov and P. K. Morokov	101	11
Moisture Content of the Gases in the Working Space of the Open-Hearth Furnace and the Hydrogen Content in the Steel. K. T. Kurochkin, B. A. Baum, A. F. Konovalov, V. V. Postyka, and S. M. Kirov	106	16
Increasing the Charge of the Bessemer Converters at the "Krivorozhstal" Works. B. E. Gurevich	109	19
The Mechanization of Labor-Consuming Operations at the Magnitorsk Metallurgical Combine A. M. Emel'yanova and V. P. Emel'yanov	111	21
ROLLED AND TUBULAR PRODUCTS		
New Technological Processes in Steel Rolling. A. I. Tselikov, E. S. Rokotyan, and E. R. Shor	113	23
Change in the Roll Design for Rolling the No. 4 Die Section. M. V. Shuraled and S. G. Nekrasov	116	26
The Production of Thick-Walled Tube Billets on Tube-Rolling Mills. A. B. Ivanov	118	28
The Mechanization of Labor-Consuming Operations at the Magnitorsk Metallurgical Combine. V. P. Emel'yanov	122	31
MISCELLANEOUS PRODUCTS		
Equipment for Entering the Rod into the Die. G. M. Fomin	124	33
The Efficiency of Steel Ropes. B. M. Vorontsov	126	34
PRODUCTION ORGANIZATION IN FACTORIES		
The Storage of Spare Parts of Equipment	127	36
A Rational Utilization of Welding Slags. G. P. Peichev and N. S. Kormilitsyn	129	37

CONTENTS (continued)

	PAGE	RUSS. PAGE
FROM THE HISTORY OF TECHNOLOGY		
The History of Furnace Technique in Russia. L. B. Gol'denberg	130	38
METALLURGY ABROAD		
A Visit to Korean Metallurgists. P. Druzhinin	132	39
NEW BOOKS	133	40

WE SHALL FULFILL THE TASKS OF THE SEVEN-YEAR PLAN AHEAD OF TIME

The Extraordinary Twenty-first Congress of the Communist Party of the Soviet Union demonstrated a great unity between our Party and the people. The Soviet people justly gave the Twenty-first Congress the name of "Congress of the Builders of Communism". It marks the entry of our country into a new historic period—the period of a large-scale development of the communist society.

The resolutions passed by the Congress open new vistas for our advance, they equip all Soviet peoples with a magnificent fighting program and illuminate the further road to communism.

The Congress considers the main tasks in the forthcoming seven-year plan to be: "in the economic field—all-round development of the productive forces of our country, the attainment of such an increase in the rate of output in all branches economy on the basis of a preferential development of heavy industry, which would allow us to make a decisive step in the establishment of material and technical bases for communism and in ensuring victory of the USSR in peaceful economic competition with the capitalist countries. A strengthening of the economic potentialities of the country, further technical progress in all branches of national economy and a continuous increase in productivity should ensure a substantial improvement in the living standards of the people."

"The Planned Figures for the Development of the National Economy of the USSR for 1959-1965", passed by the Congress, envisage a speeded-up development of the iron and steel industry and, in particular, its iron-ore bases. In 1965, it is planned to produce 65-70 million tons of pig-iron and 86-91 million tons of steel, to bring the output of rolled products up to 65-70 million tons and to mine 150-160 million tons of iron ore. The production of alloy and low-alloy steel will increase significantly, the output of electrically produced steel will increase by a factor of 1.7-2, and the production of sheet steel by a factor of 2.

The increase in the output of steel will be achieved by putting new industrial units into operation as well as by a further improvement in the utilization of existing plants and machinery and by perfecting production technology.

In 1959-1965, it is planned that new industrial units for pig-iron production amounting to 24-30 million tons, steel production — 28-36 million tons, and rolled-steel—23-29 million tons will start operating. About 100 billion rubles has been earmarked for the construction of iron and steel industrial establishments, and that amount will be 2.4 times as large as the capital investment in the iron and steel industry for 1952-1958. Capital investments for the development of the iron-ore industry will increase by a factor of 2.8.

Substantial changes in the distribution of iron and steel establishments are envisaged in the Planned Figures. It is planned that the third metallurgical center in the country, including a pig-iron output of approximately 9 million tons, will be put into operation. The share of the Eastern regions in the total output of the Union in 1965 will constitute approximately 44% in pig-iron output, 48% in steel output, and 49% in rolled-steel production.

The pig-iron and steel output in the Russian Federation, in the Ukraine and in Kazakhstan in increasing substantially.

In the course of the Seven-Year Plan, Kazakhstan will become one of the largest iron-ore bases in the country. The output capacity of the Sokolovsk-Sarbai Ore Beneficiation Combine which is being built in the Kustanai Province will exceed the sum total of the capacities of the mining industry of Persia, India, Canada, and Spain. The Karaganda Metallurgical Works will undergo substantial development and, in 1965, it will produce almost as much steel as was produced by all the metallurgical works of tsarist Russia in 1913.

Envisaging a great increase in the output of ferrous metals, the plan involves not only the building of new establishments, but also the extension and modernization of existing establishments, which should account for more than three-quarters of the planned increase in the output of pig-iron, steel and rolled-steel products. One can see from the case of the industrial establishments of the Chelyabinsk Sovnarkhoz (National Economic Council) what possibilities are at the disposal of metallurgical works. The Chelyabinsk Sovnarkhoz has worked out practical measures which, by way of the expansion, reconstruction and modernization of existing establishments, ensure an increase in steel output by a factor of more than one and a half. At the Magnitogorsk Metallurgical Combine alone, it is hoped to increase the output of rolled-steel from 5.2 to 8.5 million tons in this way.

A great potential reserve for a further increase in metal output is a more efficient use of metallurgical plants. At present, the ratio of the working volume of the blast furnace to the daily output for iron and steel regions of the country constitutes, on the average, 0.77, and the steel output per square meter of the open-hearth furnace bottom is equal to 7.55 tons. The experience of leading establishments (the Magnitogorsk Metallurgical Combine, the Nizhne-Tagil Metallurgical Combine, the "Zaporozhstal'" and the Dzerzhinskii Works) shows that

the iron and steel industry has at its disposal, substantial possibilities for achieving a considerable increase in pig-iron and steel output by means of a more efficient use of industrial plants.

What does a more efficient use of iron and steel plants mean? Calculations have shown, for instance that an improvement of only one hundredth in the ratio of blast furnace working volume to the daily output as compared with the planned value would result in an increase in pig-iron output by one million tons in 1965; and an increase in steel output per square meter of the open-hearth furnace bottom by only one-tenth of a ton will ensure an increase in the total output of steel by more than 800,000 tons.

On the basis of further technical progress, it will be necessary to introduce new, highly effective technological processes and production methods into the iron and steel industry. One of these advanced methods is the production of pig-iron and steel with the use of oxygen and natural gas. The essence of the task is to bring the percentage of pig iron made with the use of oxygen to 35-40% of the total pig iron output. The amount of steel made with the use of oxygen should increase to sixty-four million tons in seven years, as compared with 13.9 million tons in 1958. On account of the use of oxygen, the plan envisages an output of more than 3 million tons of pig iron and 8 million tons of steel with the existing plants in the course of the Seven-Year Plan.

The Soviet people have accepted the Seven-Year Plan as their own, intimate, vital task. In the pre-Congress competitions, the iron and steel workers had already written many glorious pages. And now too the iron and steel workers are in the forefront of the productivity competition. The personnel of the largest metallurgical establishments — the Kuznetsk and the Magnitogorsk Metallurgical Combines, and of other works — in response to the resolutions of the Twenty-first Congress of the CPSU, have revised their obligations and decided to fulfill the 1959 Plan ahead of time.

"For the sake of the great cause of building communism, one can and should work hard". Armed with the resolutions of the Twenty-first Congress of the CPSU, Soviet iron and steel workers are mobilizing all their forces for a successful realization of the Seven-Year Plan for the development of national economy and they will succeed in fulfilling the 1959 Plan — the first year of the Seven-Year Plan — ahead of time.

IMPROVED PREPARATION OF SMALL COKE

V. V. Vinogradov

Mundybash Sinter Plant

According to the standard technological instructions on sintering iron ores, small coke of a size not larger than 25 mm, received at the sinter plant, should be crushed in a roller crusher in order to obtain the final product containing not less than 90% of 3-0 mm fraction. When the Mundybash Plant was supplied with small coke, 15-0 mm in size, the two existing standard four-roll crushers easily provided the full requirements in crushed coke containing about 91 to 92% of 3-0 mm fraction for sinter machines. In connection with the shortage of coke fines, the plant began to be supplied (up to 10-15% of total consumption) with small coke, 25-40 mm in size. This created several difficulties. The upper rolls of the crusher would not grip the larger-size fraction of the small coke and, therefore, it was necessary to increase the gap between the rolls and then to weld ribs on the rolls. In addition, when the 25-40 mm coke alone was supplied, the throughput of the crushers dropped by half, and, hence, it was necessary to increase the gap between the lower pair of rolls too. At the same time, owing to the conditions which obtained at the delivery and storage of 25-40 mm coke, it was not possible to mix it uniformly with coke fines.

All this caused a sharp reduction of the content of 3-0 mm fraction in the final product (Table 1).

The monthly average content of 3-0 mm fraction in crushed small coke constituted 82-84%. Since the Mundybash Plant produces high fluxed sinter (basicity 1.4) from charge containing up to 60% finely ground concentrate (0.3-0.0 mm), the introduction of poorly prepared small coke caused a sharp deterioration in the quality of the sinter with regard to its sulfur content, mechanical strength and lumpiness.

TABLE 1. The Effect of the Content of 25-40 mm Fraction in the Initial Small Coke on the Content of 3-0 mm Fraction in the Crushed Product

Content of 25-40 mm fraction in initial small coke, %	Content of 3-0 mm fraction in crushed coke, %
10	85.0—89.2
20	77.5—82.1
50	66.0—75.7
100	49.2—55.6

With view to improving the efficiency of the size reduction of the small coke, it was decided to introduce a preliminary crushing of coke (project put forward by Engineer G. I. Igoshin). For the preliminary crushing of coke, the hammer mill was chosen since it does not crush the coke too finely and has a high throughput. The experience on the operation of the Mundybash Plant shows that, if the content of 0.5-0 mm fraction in the coke is high, the conditions of coke combustion deteriorate and the quality of the sinter falls.

The hammer mill was made at the machine shop of the Plant. The characteristics of the mill are: rotor diameter — 980 mm, rotor length — 1000 mm, speed —

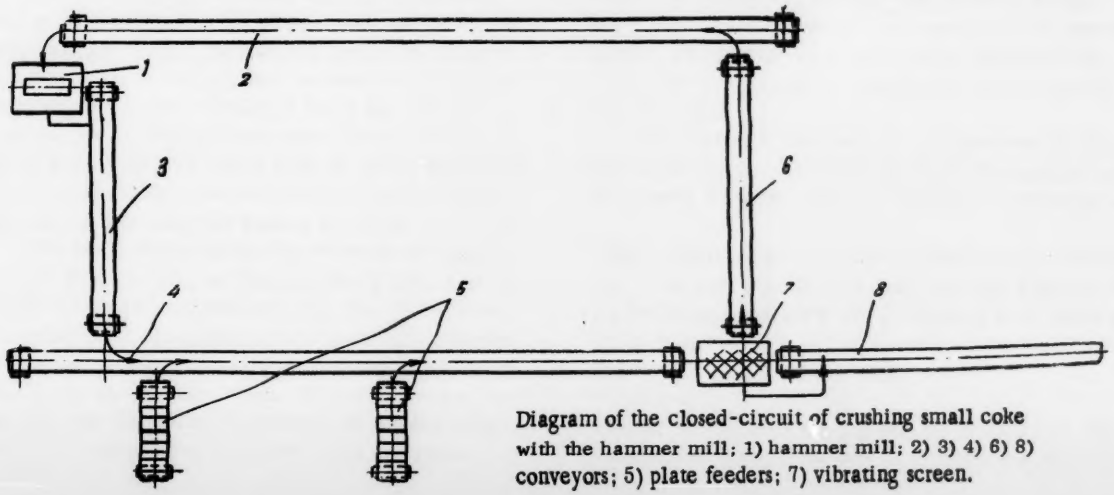


Diagram of the closed-circuit of crushing small coke with the hammer mill; 1) hammer mill; 2) 3) 4) 6) 8) conveyors; 5) plate feeders; 7) vibrating screen.

575 rpm, discharge openings between the screen bars – 25 mm, motor – 100 kw, maximum throughput – approx. 50 tons/hr. A GUP-1 vibrating screen 1250 × 3500 mm in size was included in the closed circuit with the crusher. Small coke was delivered by four conveyor belts, 650 mm wide and 99 m long. The diagram of the circuit for the preliminary crushing of small coke is shown in the figure.

Table 2 shows the screen analysis of the product of crushing and screening the small coke when the initial small coke contains 20% of 25-40 mm fraction and the openings of the screen are 19 mm in size.

In 1.5 months of operation with this scheme, the content of 3-0 mm fraction in the final product constituted on the average 94.5%; at the same time in none of the single analyses did the content of 3-0 mm fraction exceed 90%. The content of 0.5-0 mm fraction in the final product in the two-stage crushing operation even decreased. The improved crushing of coke together with some other measures made it possible to decrease the mechanical strength of the sinter and to reduce its sulfur content (Table 3).

Consequently, the two-stage crushing process with the use of the hammer mill makes it possible to improve the size reduction of coke considerably and, hence, to improve the quality of the sinter. Moreover, in this way it is possible to achieve the 2-0 mm or even 1.2-0 mm size of coke and this is especially important for obtaining a high-basicity sinter from finely ground concentrates.

It should be pointed out that during the operation of the hammer mill a great amount of dust is produced. The dust, however, can easily be eliminated by the sealing of the machinery in the installation of extraction equipment and the application of steam at places of dust escape. The undisputed advantages of the hammer crusher are its high output and the absence of excessively reduced particles.

TABLE 2. Results of the Introduction of a Two-Stage Crushing of Coke

Location of sampling	Fraction content (%), mm:					
	40-25	25-12	12-6	6-3	3-0.5	0.5-0
In front of the hammer mill	90.0	9.6	0.4			
After the hammer mill	8.5	47.8	16.1	7.5	10.5	9.6
Undersize product	–	1.7	14.1	21.6	48.2	14.4
After four-roll crusher	–	–	–	4.7	60.8	34.5

TABLE 3. Dependence of the Quality of the Sinter (according to the screen analysis of the product dispatched from the storage) on the Method of Crushing the Coke.

Sulfur content, %	Amount of sinter, % of total output		Yield of 5-0 mm fraction after trommel test, %	Amount of sinter, % of total output	
	With one-stage coke crushing	With two-stage coke crushing		With one-stage coke crushing	With two-stage coke crushing
0.04	1.3	2.1	below 29.0	–	2.9
0.05	22.6	32.6	29.1 - 30.0	3.2	11.1
0.06	46.9	53.3	30.1 - 31.0	12.3	37.8
0.07	22.3	11.4	31.1 - 32.0	59.3	47.6
0.08	3.6	0.6	32.1 - 32.5	19.8	0.6
0.09	2.7	–	above 32.5	5.4	–
0.10	0.3	–	–	–	–
above 0.10	0.3	–	–	–	–

Note. The content of –10 mm fraction in the sinter dispatched decreased from 31.4% to 30.6%.

THE AUTOMATIC CONTROL OF THE COMPOSITION AND TEMPERATURE OF GAS OVER THE HORIZONTAL CROSS-SECTION OF THE BLAST FURNACE

Z. I. Nekrasov and Ya. M. Obodan

Institute of Ferrous Metallurgy, Acad. Sci. Ukr. SSR

For correct operation of the blast-furnace process it is necessary to have, as well as data on the stock level, also information on the composition and temperature of the gas in the cross section of the furnace directly under the stock line where the gas has already completed, in the main, its reducing and thermal action. Below, we give a description of an apparatus, developed by the Institute of Ferrous Metallurgy, Acad. Sci. Ukr. SSR, and employed at several blast furnaces, for the automatic control of the composition and temperature of the gas along the diameter of the furnace*.

The apparatus consists of two cantilever beams (one long and one short), an automatic self-recording gas analyser, a self-recording electronic potentiometer, a change-over switch for sampling points, a thermocouple and gas and pulse lines.

Each of the beams (Fig. 1) consists of a hollow, welded (or cast) water-cooled enclosure, inside which gas intake tubes are located for sampling gas and stabilizing its composition at various points on the furnace diameter, as well as for mounting the thermocouples. Gas from the furnace passes continuously from the beam through $\frac{1}{2}$" diameter tubes to the control room. Here, part of the gas from the gas line is intermittently sucked off by the automatic gas analyzer, and the excess gas passes through a liquid seal and a flare into the atmosphere

Fig. 2 shows how the sampling beams are placed at the blast furnace.

Gas-intake tubes and filters used are of the same type as in the apparatus for sampling and analyzing the peripheral gas†. If a gas analyzer is attached to each sampling point of the gas, one can ensure a continuous recording of the gas composition along the furnace diameter for all intermediate points simultaneously. At the same time, one has to study recording charts from several instruments in order to form an opinion of the gas composition along the furnace diameter.

The graph of the carbon dioxide content along the furnace diameter may be obtained on a single chart by means of one gas analyzer only (Fig. 3). In this case, the switch-over of sampling points to one gas analyzer is effected automatically by means of the switch valve used at the Dzerzhinskii Works. Recording the gas composition over the whole diameter of the furnace takes 1-1.5 hr, depending on the speed of the analyses‡.

The advantage of the switching is the fact that all the data can be recorded on one chart. It is seen on the specimen chart (Fig. 4) that after three analyses of air ("zero setting" test), the apparatus records three analyses from the peripheral point "a". Then recordings are made in consecutive order of three analyses of each sample, taken from the series of intermediate points "b", "c", "d", "e" in the direction away from the wall, and from point "f" at the furnace center. Then, the gas analyzer is connected to the sampling points located symmetrically on the opposite side of the furnace center in the other cantilever beam, i.e., points "g' ", "h' ", "i' ", "j' ", "k' ". This cycle of operations is repeated automatically at equal time intervals.

If it is necessary to have a continuous check on the gas composition in any horizontal cross section of the furnace (for example, when determining optimum burdening, during irregular operation of the furnace, etc.), it is preferable to connect each sampling point of the beam to a separate gas analyzer.

During a steady operation of the furnace, when the gas composition in each point does not change significantly with time, the beams are connected to the automatic switch. At works in the South, "Stakhanovets"-type gas analyzers are used.

The outer ends of the gas-intake tubes in the cantilever beams are provided with packing seals for tightening the thermocouples employed for measuring the temperature of the medium at the sampling points of the gas along the furnace diameter. The readings of the thermocouples are recorded on the electronic potentiometer chart. For this purpose, it is desirable to have rapidly moving charts.

The apparatus described is complementary to the equipment for the automatic testing of the composition of the peripheral gas. The availability of continuous

*The apparatus was proposed by Corresponding Member of the Acad. Sci. Ukr SSR, Z.E. Nekrasov and Engineer, V. I. Listopadov. Engineers Ya. M. Obodan, I. K. Kotov and V. V. Shcherba took part in the work on the realization and improvement of the project.

†Metallurgist No. 2 (1959).

‡As the technique of the gas analysis is improved, the time period of the cycle may be reduced to a few minutes.

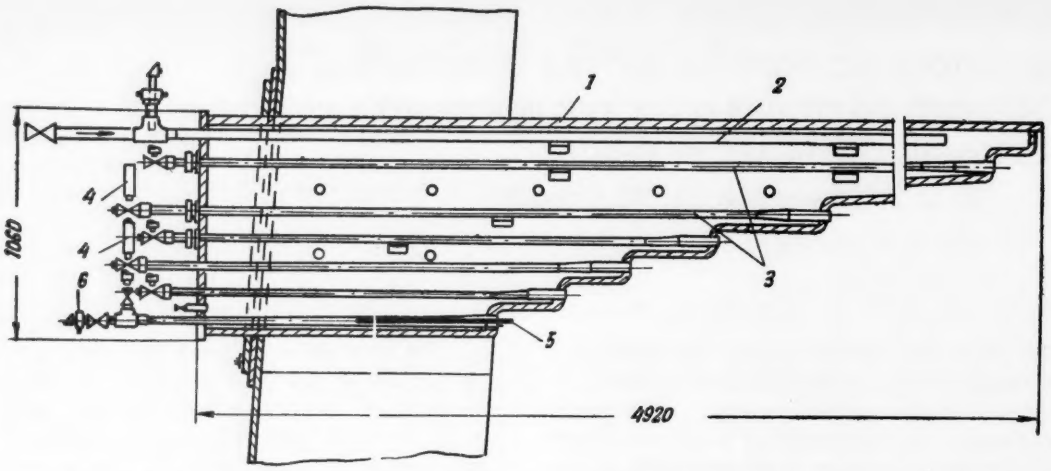
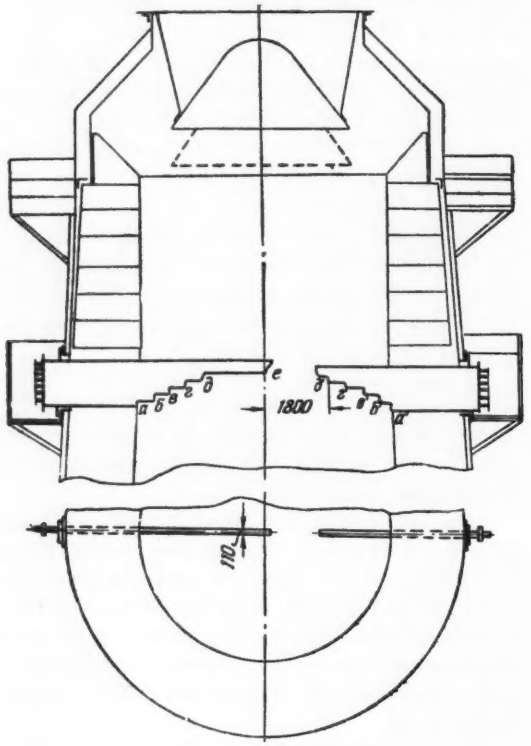


Fig. 1. Long cantilever beam: 1) beam jacket; 2) water feed tube; 3) gas intake tubes with stabilizers; 4) filters; 5) thermocouple; 6) packing seal.



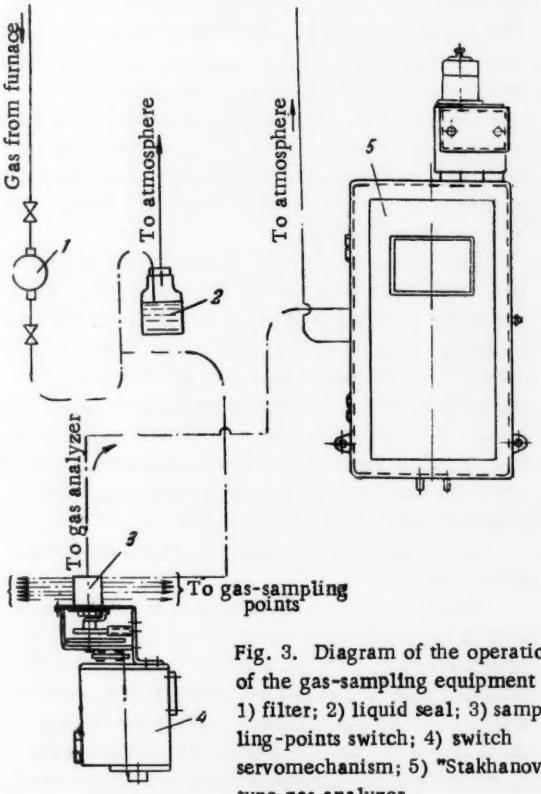


Fig. 3. Diagram of the operation of the gas-sampling equipment: 1) filter; 2) liquid seal; 3) sampling-points switch; 4) switch servomechanism; 5) "Stakhanovets"-type gas analyzer.

data on the composition and temperature of the gas over the cross section of the furnace assists in a rapid detection of any deviation from normal operating conditions (deviations from optimum distribution of charge materials and gas stream, and from normal operation of the charging equipment; deterioration in the blast and slag regimes; changes in the condition of the furnace contour). In addition, when the charge composition is changed, the automatic control makes the choice of the best charging method easier.

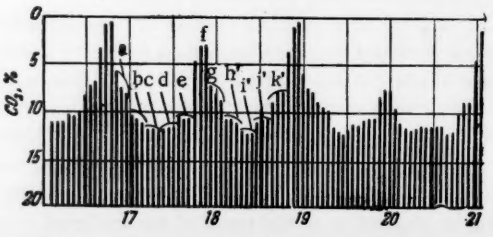


Fig. 4. Chart showing the gas composition along the diameter of the furnace.

CHARGING EQUIPMENT WITH DOUBLE-WALL CONE

I. L. Kordabnev

Dzerzhinskii Factory

When the pressure at the blast-furnace top was increased at our factory, the service life of the charging equipment decreased from 3-5 years to 9-13 months. The cause of such a short life of the equipment was the wear of the contact surfaces of the bells and cups under the abrasive action of impure blast-furnace gas.

The rate of the wear depends on the amount of blast-furnace gas carryout and the pressure of the blast-furnace gas. Various measures were taken to eliminate this deficiency: all-cast "flexible" cups and "rigid" bells were made; the openings for stock indicators were accomodated outside the cup or placed in the cup flange, so that the cup could have the accurate shape of a body of revolution; the casting technique of cones and sealing cups was improved; the contacting surfaces were plated (welded-on) with hard alloys and machined so that the welded-on layer would be strong enough and not affected by the temperature fluctuation and impacts of the cone against the cup, and initial gaps between the cone and the cup would not exceed 0.1 mm. The cone surface, up to a height of 1000 mm above the contacting surface, was plated with a hard alloy to prevent wear, and then its surface was ground up to 300-400 mm of its height to avoid a breakdown of the cup under the impacts of the cone during the entry of the charges. To prevent a breakdown of the contacting surface of the cup under the shocks of the cone, the speed of closing the large cone during the entry of the charge was reduced, cooling of the cup was applied, the operation of the equalizing values was modified and the intercone space was filled with semi-purified gas after each lowering of the small bell.

The revolving distributors were provided with double stuffing boxes, the height of the stuffing boxes and the number of lubrication points on the lower stuffing box were increased and a complex gas-tightening of the rods (stuffing box, labyrinth and steam) was applied. In addition, the lower sleeve of the small cone rod were made with varying-length ends in order to prevent the wear of the large cones down to a dangerous limit. An all-cast, thick-wall hopper of the distributor with removable lower ring consisting of two halves, was also used.

All these measures, however, did not ensure a reliable and long operation of the charging equipment. In spite of some extension of the service life of the charging equipment its durability again decreased with the pressure at the furnace top was recently increased to 0,8 atmos.

Thus, the charging equipment replaced as late as 1958, lasted for 10-13 months. Such a frequent replacement of the charging equipment does not improve the position, and so the furnaces are operating some of the time with worn parts of the charging equipment and this adversely affects the productivity of the blast-furnace shop. In addition, the low durability of the charging equipment results in an increase in the idling time of the blast furnaces and does not permit the pressure of the gases at the furnace top to be increased to 1.5 atmos.

Recently, accounts appeared in the literature showing that the application of automatic welding-on (plating), under a layer of a flux, of the contacting surfaces of the cone and the cup makes it possible to increase the service life of the charging equipment to 4-5 years. Nevertheless, operational experience shows that the quality of the plating, although constituting one of the conditions for the normal operation of the charging equipment, cannot, by itself, increase the life of the charging equipment significantly. Therefore, the following suggestions, put forward recently, regarding an improvement in the durability of the charging equipment were investigated at the Dzerzhinskii Factory:

1) an increase in the angle of the cone in its lowest part to 60-62° (Fig. 1);

2) a supply of steam or gas to the ring grooves in the cup;

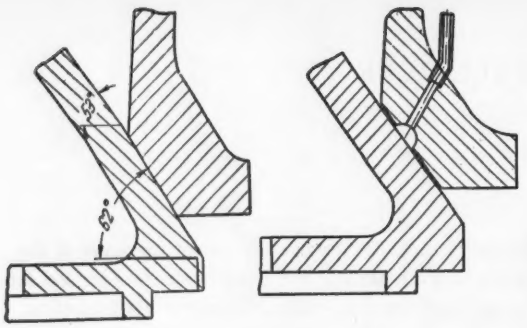
3) three-cone charging equipment;

4) charging equipment with double-wall cup (Fig.2), proposed by the author of this article.

As a result of the analysis of all these proposals it was decided to try out the three-cone charging equipment and the equipment with double-wall cups. The three-cone equipment is now being made, and the equipment with double-wall cup was set up on one of our blast furnaces at our factory in May, 1958.

The principle of the operation of this equipment is based on filling the space between the walls of the cup with partly purified gas supplied under pressure from the gas main. Owing to the excess pressure set up between the cup walls, the abrasive action of the impure blast furnace gas on the cone and the cup is reduced.

The objections against the adoption of the double-wall cup boiled down to the argument that such a cup is difficult to manufacture and that it is altogether impossible to tighten the two contacting surfaces of the cup against the contacting surface of the cone. Experience, however, on making and operating the charging equipment proved these objections to be completely unfounded. There was no difficulty in making the double-wall cup; its welded construction was adequately strong and reliable. However, in the design andproduction of the first charging



a b

Fig. 1. Sealing the gap between the cone and the cup: a) by increasing the cone angle; b) by means of steam or gas.

equipment with a double-walled cup, some very serious errors were made.

The temperature difference between the internal and external sides of the cup was not taken fully into account in the design. Owing to an unfortunate change in the technique of plating, the contacting surfaces of the cup were not plated properly. When they were machined, it was found that they were covered with numerous cavities distributed uniformly on the circumference and elongated along the generatrix from one edge of the contacting surface to the other. The cone intended for the new charging equipment was not ready on time and, therefore, it was necessary to adapt the large cone of the ordinary charging equipment. Therefore, it was necessary to machine-off the plating of the lower contacting surface almost completely, and to machine the upper contacting surface inadequately (some rough places were left). But even under those conditions the gap between the internal wall of the cup and the cone was found to be 1.2 mm.

It would appear that all these deficiencies must inevitably result in the failure of the experiment. They, however, provided an extra proof of the soundness of the idea of the double-wall cup. The gap between the contacting surfaces of the internal wall of the cup and the cone decreased considerably even during the initial stages of operation owing to the elastic deformation of the external wall of the cup under the action of the pressure of the gases onto the cone, and then, as a result of the permanent deformation of the external wall of the cup, the gap became even smaller.

The leakage of semipurified gas, taking place in the initial period of operation, has no harmful effect, since semipurified gas has no abrasive properties. One should not, however, ignore the harmful effects of the poor-quality plating of the contacting surfaces on the operation of the charging equipment. An ordinary piece of equipment with such poor-quality plating would be rendered useless much sooner than one with proper plating. This is confirmed by the fact that the charging equipment of one of our blast furnaces was worn out in the course of the first four months of operation, although the plating of its contacting surfaces was somewhat better. The charging equipment with the double wall cup will probably operate for a somewhat shorter time than expected in the design owing to the poor plating. Even so, there will be adequate proof of its advantages over other designs.

At present, after eight months of operation, the charging equipment with the double-wall cup is in excellent condition. Daily checks (by means of instruments and direct observation) show that the leakage of impure gas between the contancting surfaces of the external wall of the cup and the cone are almost nonexistent even when the semipurified gas is cut off, because, owing to the presence of two contacting surfaces with a considerable distance between them, the cone cannot get out of alignment when being closed and alwasy adheres tightly to the cup over the whole circumference.

The leakage of semipurified gas between the contacting surfaces of the internal walls of the cup and the cone is insignificant (in any case, it does not exceed the leakage of impure gas in the ordinary charging equipment in good condition), However, in an ordinary charging equipment, the passage of impure gas causes considerable

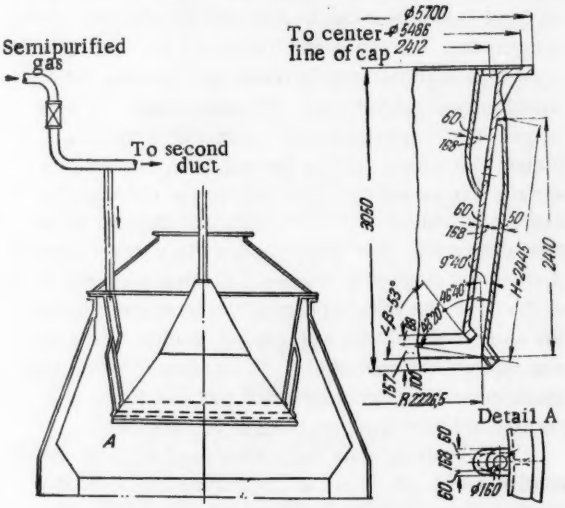


Fig. 2. Charging equipment comprising double-wall cup.

erosion which leads to a breakdown of the equipment, whereas the semipurified gas has no harmful effect on the contacting surfaces of the cone and the cup. This fact is confirmed by the long life of the charging equipment at factories where there is small blast-furnace dust carryout (the Magnitogorsk Metallurgical Combine, Kuznetsk Metallurgical Combine, and others), and where the service life of the charging equipment is up to 2-2.5 years. However, even with the small dust carry out at those works, the charging equipment finally wears out because of this erosion.

The introduction of the equipment with the double-wall cup makes it possible to increase the service life

of the small cone. Up till now, the service life of the small cones at the blast furnaces of our factory constituted 4-5 months (during the initial period after the replacement of the charging equipment, the service life of the small cones was up to 7-9 months, and later on, when the condition of the charging equipment deteriorated, their service life decreased to 2-3 months).

Small cones usually became unusable as a result of the erosion of their contacting surfaces by the impure gas. The adoption of the double-wall cup eliminates this cause of erosion, and the wear of the small cones is due to the abrasive action of the charge materials. Therefore, it is necessary to armor-plate the small bells, as, in this way, the service life will be increased to 1.0-1.5 years.

The adoption of the double-wall cup will increase the life of the charging equipment, will make a reduction in the idling time of the blast furnaces and a change-over to operation at a gas pressure at the furnace top of 1.5 atmos possible.

Editor's Note. Increasing the life of the charging equipment at blast furnaces working at a high pressure in the furnace top is an extremely important problem. The double-wall cup, proposed by I. L. Kordabnev, is a very original solution for this problem. However, it will be possible to draw final conclusions after more operational experience.

We shall be grateful for readers' views on the problem raised.

REDUCING THE SULFUR CONTENT IN PIG IRON

P. M. Matveev

Novolipetsk Metallurgical Factory

The desulfurization of pig iron in the blast furnace proceeds according to the reversible reaction

$$FeS + CaO \rightleftharpoons CaS + FeO.$$

The higher the content of FeO in the slag, the slower is the desulfurization process. As is known, part of the iron is oxidized to ferrous iron in the deoxidizing zone and is afterwards reduced in the hearth. However, the conditions for the reduction of the ferrous oxide are not always favorable. If, after a prolonged operation with high-basicity slags, the walls of the hearth become covered with deposits of lime materials and the volume of the molten-iron (hearth) decreased, the desulfurization reaction proceeds in the reverse direction until equilibrium is attained, and thus the sulfur content of the pig iron increases.

This fact is confirmed by operational experience at our factory. For comparison, we have taken the operating data of No. 1 blast furnace for November, 1955, when the hearth walls became covered with slag deposits; and for May, 1956, when the volume of the hearth was approximately equal to the design volume.

	November 1955	May 1956
Ratio of the number of pig-iron tappings with a small quantity of slag to the total number of tappings, %	15.2	4.3
Amount of pig iron downgraded because of high sulfur content, tons	6695	4705
Including amount downgraded because the slag basicity ($CaO : SiO_2$) had been decreased to 1.15 or less, tons	806	3160
Average monthly basicity of the slag	1.24	1.22

The extent to which the hearth is "overgrown" with lime deposits is determined by the ratio of the number of the pig iron tappings with a small quantity (up to 1.75 ladle) of the slag to the total number of tappings.

It is seen from the data quoted that, when the walls of the hearth are not overgrown, the amount of pig iron which does not conform to the technical specifications is considerably lower and a large part of this pig iron is substandard owing to an excessively low slag basicity.

In order to decrease the sulfur content in pig iron, one should "wash" the hearth with Bessemer slags or other acid materials which dissolve the lime-material deposits on the hearth walls and, hence, the volume of the hearth is increased. During the washing, it is necessary to increase the coke input slightly. A temporary lowering of the slag basicity which takes place during the washing operation has no marked effect on the sulfur content in the pig iron.

One can estimate the extent of the deposition of lime materials on the hearth walls by the number of ladles of slag collected furing the iron tapping. If the volume of the slag constitutes 1-1.5 ladles, the hearth is overgrown and one should start to wash it. Under the conditions which obtain at the Novolipets Factory for the washing operation, one introduces Bessemer slag into the charge during the hot operation of the furnace to the amount of 3 tons per round. The hearth is washed 3-4 times until all the deposits in the hearth are removed.

The ratio of the amount of the upper slag to the lower slag should be 0.6 : 0.4. A reduction in the amount of the lower slag will result in an increase in the sulfur content in the pig iron.

EXPERIENCE ON THE APPLICATION OF OPEN-HEARTH SINTER

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Kuznetsk Metallurgical Combine

Until now, the Kuznetsk Combine has used Magnitogorsk open-hearth ore. A considerable reduction in the supplies of this ore for the open-hearth plants has been envisaged for 1959. Therefore, the problem of the use of a substitute for open-hearth ores – a specially prepared open-hearth sinter – becomes of great importance. As is known, the sintering process, side by side with the beneficiation, opens unlimited possibilities in the utilization of local iron ore supplies.

In 1949-1951, the "Mekhanobr" Institute developed a method of producing heavy open-hearth sinter of a weight of 3.2-3.3 tons / m^3 by means of adding cast-iron turnings to the charge materials. Experiments on this sinter have shown convincingly that it is quite possible to use the sinter in open-hearth production. This is also confirmed by operational experience at the Nizhne-Tagil and Magnitogorsk Combines, the "Azovstal'" Factory and other factories.

It was established at those works that open-hearth sinter should have a high content of iron and a low content of silicon, phosphorus and sulfur; the sinter should consist of large-size particles, it should be dense and strong and have a high oxidizing power.

Two experimental batches of sinter were prepared for the steelmaking shops of the Kuznetsk Combine; one (1400 tons) at the Abagursk Sinter Plant and one (650 tons) at the Mundybash Sinter Plant from Tashtagol iron ore with the addition of 15-18% of rolling-mill scale. A comparison of the chemical composition and physical properties of the sinter and iron ore is given in Tables 1 and 2.

The experimental sinter had large-pore structure and, therefore, it had a low mechanical strength and high brittleness which was conducive to its breaking down into small particles during unloading. The weight per unit volume of the sinter at the sinter plant constituted 1.45-1.50 tons / m^3, but before it was charged into the furnace the weight was 1.70-1.75 tons 1 m^3, as a result of size reduction during unloading operations. Owing to the high content of iron oxides, the oxidizing power (determined by the total content of oxygen in the sinter) was only slightly lower than the oxidizing power of iron ore.

Rimmed, high-carbon killed, and alloy steels were made with the use of the open-hearth sinter in 380-ton and 190-ton furnaces. In all, 35 experimental heats were carrie d out. The sinter was added during the charging and working periods; the proportioning of the charge was not changed. The following order of charging loose materials was accepted. When iron ore was fully replaced by the sinter, the sinter was introduced in the amount of two charging boxes through each charging door, and then limestone and the remaining sinter was charged layer by layer. When only part of the ore was replaced (60% sinter), two charging boxes of the sinter were charged through each charging door, and then lime and the remaining sinter and ore were added. The sinter was not heated, but each layer of limestone was heated for 5-7 min. No bauxite was added to the charge.

When the sinter is charged directly onto the bottom, no vigorous reactions involving metal and slag eruptions take place, and the losses of the sinter in the slag become smaller.

When the full amount of ore which was to be introduced during the charging period was replaced by sinter (when 3 kp steel was made) the consumptiom of sinter did not exceed that of the ore (Table 3) and consitituted, on the average, 14% by weight of accepted ingots (the average consumption of Magnitogorsk ore was 14.05%). In the production of rail steel, the sinter

TABLE 1. The Chemical Composition of Sinter and Iron Ore

	Chemical composition, %									
	Fe_{total}	FeO	Fe_2O_3	SiO_2	Al_2O_3	CaO	MgO	MnO	P	S
Abagursk sinter	61.22	18.46	67.05	7.32	2.30	3.32	0.36	0.44	0.07	0.04
Mundybash sinter	58.02	28.58	54.10	6.82	2.12	3.31	2.59	0.69	0.07	0.09
Magnitogorsk iron ore	60.27	10.92	74.06	7.40	2.51	0.99	0.16	0.17	0.07	0.05
Tashtagol iron ore	48.00	—	—	13.0	5.92	6.26	0.92	0.92	0.24	0.03

consumption exceeded that of iron ore by 0.9% and amounted to 11.6% on the average. In the production of alloy steels in 190-ton furnaces, the amount of sinter in the charge was also higher than that of iron ore on average by 0.3% of the weight of accepted ingots.

TABLE 2. The Physical Properties of the Sinter and Iron Ore

	Basicity $\frac{CaO}{SiO_2}$	Mechanical strength, kg/cm²	Melting temperature, deg C	Weight per cubic meter, tons/m³	Specific weight, g/cm³	Moisture content, %	particle size %	
							below 15 mm	above 50 mm
Abagursk sinter .	0.45	90	1490	1.75	4.51	Het	20	15
Mundybash sinter	0.48	92	1530	1.73	4.25	Het	27	13
Magnitogorsk iron ore	0.13	450	1550	1.85	4.23	2.5	2	70
Tashtagol iron ore	0.48	360	1540	1.80	4.15	3.0	12	23

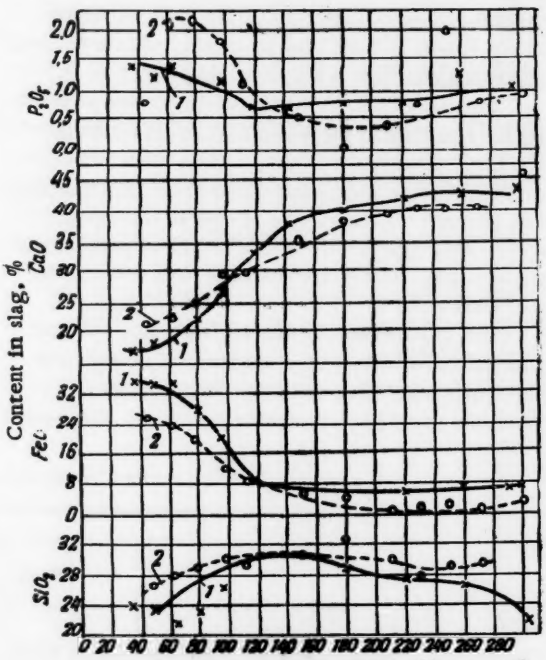
The input of limestone during the charging period in the production of steel of all grades was lower in the heats in which the sinter was used (it was lower by 0.3% for rail steel, 1.26% for alloy steel, and 0.5% for 3kp steel). The general increase in limestone consumption during the production of steel with the use of iron ore is explained by the fact that the basicity of iron ore is approximately 3.5 times lower than the basicity of sinter.

The first experimental heats with the use of sinter showed that, at the same input of limestone as before, the melting proceeds in the presence of more viscous slag. The basicity of the slag was approaching 2.8-3.0, and, therefore, the input of limestone was afterwards reduced by 2-3 tons without any deterioration in the process of slag formation. The diagram shows that, when sinter is used the process of slag formation during the melting period an the dephosphorization proceeds in the same way as in the case of ore. However, at the time of the melt-down of the charge the phosphorus content in the metal is, on the average, 0.011% lower than in the heats in which ore is used, the sulfur content being the same (0.040-0.045%). This is explained by the fact that when sinter is used, the removal of slag proceeds at a higher rate, the slag is more fluid and it is removed more easily and to a greater extent from the furnace and, therefore, the conditions for the dephosphorization are much more favorable. The composition of the slag during the various periods of the heats was approximately the same for the heats with ore and with sinter. Also the action at the slag on the bottom and the banks of the furnace was the same in both cases. The analyses of the slags at the various periods of the heat shows that the amount of MgO in the slag was low and of the same order for both types of heat (8-10%).

When sinter is used, one of the very important factors in a reduction in the duration of the heat and its separate periods. When 3kp steel was made, charging the loose materials for the heat in which sinter was used took on the average, one hour 19 min, and for the heat in which ore was used it took one hour 20 min, i.e. the duration of the charging operation was practically the same. However, when sinter was used in the heat of rail steel, the charging of the loose materials was reduced to 37-58 min.

The analysis of the results of the experimental heats shows that the optimum lengths of time of heating the charge for the heats with the sinter is 1 hr 35 min to 1 hr 45 min, and for the heats with the iron ore it is 10 to 15 min more. In the production of steel with the use of sinter, the prolonged charging period and heating period (over 45-50 min) of the loose materials sharply increase the duration of the process. When iron ore is used, one may consider an optimum duration of the charging and heating periods for loose materials to be 50-70 min. If the charging and heating periods last longer (more than 75 min) the melt-down period increases sharply and, consequently, so does the whole duration of the heat.

An analysis of the experimental results showed that the duration of the charging and heating periods of the



Variation of slag composition in the course of the heat: 1) Heats in which sinter was used; 2) heats in which ore was used.

TABLE 3. Comparative Data on the Heats with Sinter and with Iron Ore

Property	3kp rimmed steel		R-50 rail steel		20Kh-45Kh alloy steel	
	with sinter	with iron ore	with sinter	with iron ore	with sinter	with iron ore
Number of heats	8	11	5	10	5	6
Charge composition, %:						
hot metal	58.8	59.3	59.8	59.7	62.6	60 8
scrap	41.2	40.4	40.2	40.3	37.4	39.2
Introduced during the charging period, %						
sinter	14.0	—	11.6	—	10.9	—
iron ore	—	14.05	—	10.7	—	10.6
limestone	3.59	4.13	3.76	4.05	4.51	5.77
Content in hot metal, %:						
Si	0.78	0.78	0.73	0.79	0.78	0.77
P	0.126	0.130	0.126	0.130	0.128	0.127
S	0.042	0.045	0.042	0.044	0.042	0.044
Amount of run-off primary slags, ladles	1.94	1.77	1.77	1.72	1.27	1.10
Content in the metal after the melt-down period, %:						
C	0.69	0.61	1.05	1.11	1.20	1.16
P	0.024	0.035	0.024	0.035	0.015	0.019-
S	0.042	0.045	0.042	0.043	0.042	0.042
Content in the slag after the melt-down period, %:						
FeO	6.6	5.1	6.4	5.4	8.6	7.6
P_2O_5	1.48	0.82	1.72	1.37	1.37	1.39
Basicity of the slag after the melt-down period, CaO : SiO_2	1.84	2.05	2.32	2.46	2.07	2,01
Input during the finishing period, tons:						
ore	8.85	6.7	7.35	7.30	5.30	5.30
lime	1.35	1.90	1.88	2.92	4.90	4.25
bauxite	1.69	2.20	2.00	2.60	3.30	3.28
Rate of removal of carbon in burning, % C/hr :						
during the dephosphorization period	0.37	0.31	0.30	0.26	0.57	0.51
during the pure-boil period	0.19	0.17	0.18	0.18	0.29	0.31
Content in the metal before deoxiation, %						
P	0.014	0.015	0.011	0.014	0.011	0.011
S	0.034	0.038	0.036	0.036	0.026	0.028
Content in the slag before deoxidation, %						
FeO	7.9	8.0	8.1	8.7	8.3	8.0
P_2O_5	1.24	1.34	1.52	1.84	0.75	0.75
Basicity of the slag before deoxidation, CaO SiO_2	2.60	2.52	2.72	2.75	2.86	2.88
Duration, hr-min						
fettling	0—14	0—15	0—10	0—11	0—17	0—17
charging	3—08	2—37	2—40	2—35	1—33	1—29
heating	1—35	1—44	1—49	1—51	1—18	1—25
melt-down	4—21	4—41	3—24	3—59	2—18	2—21
finishing	1—55	1—54	2—03	2—15	2—57	2—32
total heat	11—41	11—39	10—48	11—38	8—57	8—38

loose materials have a significant effect on the phosphorus content (p.021%) in the metal after the melt-down. The minimum phosphorus content (0.021%) in the metal after the melt-down is obtained when the charging and heating periods are within the range of 60-75 min. If charging and heating the loose materials takes a shorter time, the phosphorus content in the metal after the melt-down increases to 0.025-0.026%. However, this insignificant increase in the phosphorus content in the metal should not be treated as a decisive factor against cutting down the duration of the heat.

The analysis of the data in Table 3 shows that, owing to the porosity of the sinter and its ease of melting (melting temperature of sinter is 30-60°C lower than the melting temperature of iron ore), the average duration of the melt-down period during the production of 3kp steel decreased by 20 min, and during the production of rail steel, grade R-50, it decreased by 35 min.

However, the total duration of the experimental heats of 3kp steel remained the same as before owing to the production difficulties encountered; the duration of the heat of rail steel decreased on the average by 50 min. In the production of alloy steel with the use of sinter in the 190-ton furnaces, the duration of the heat increased on the average by 19 min owing to delays during the charging period and a longer (by 25 min) working period of the heats. The heating and melting periods, however, were shorter.

Contrary to expectation when sinter was used the slag was not foamy. All doubts with regard to the durability of the roof and the checkers proved to be unfounded too.

As compared with the heats in which iron ore was used, the input of lime and bauxite during the working period was lower for the heats in which sinter was used. The basicity of the slag and the phosphorus content in the metal before the deoxidation were approximately the same for both types of heat. This is explained by the fact that the solid deoxidizing agent added during the charging period has little effect in the further couse of the working period. The additions of iron ore, lime and bauxite during the working period greatly affect the composition of the metal and slag. The rates of carbon loss in burning, during the period of the dephosphorization and pure-boil, fluctuated but slightly since both types of heat were refined with iron ore.

To determine the effect of the introduction of sinter during the working period, four experimental heats of 3kp steel were carried out in 300-ton furnaces; these heats showed that when sinter was used for the refining, the dephosphorization of the metal did not deteriorate. Before the deoxidation, the steel contained 0.013% phosphorus, on the average; and in the heats which were refined with ore it contained 0.015%. The rate of carbon loss in burning did not decrease. Thus, when sinter was used for the refining, the rate of carbon loss in burning during the dephosphorization period was 0.43% C per hour, and during the pure boil period it was 0.18% C per hour, whereas when ore was used for the refining, the average rate was 0.31% and 0.17% C per hour, respectively.

When the heat is refined with iron ore the maximum rate of carbon loss in burning takes place in the first 10-12 min, whereas, owing to the large quantity of fines when sinter is used for the refining, the increase in the rate of carbon loss in burning takes place somewhat later, as the metal is heated up. If the temperature of the metal is not sufficiently high, the sinter does not react at once with the carbon. But as the metal is heated up, the rate of carbon loss in burning increases by the beginning of the deoxidation period and that may result in the production of inferior steel. When sinter is added during the working period, the slag should have a normal basicity and contain as little undissolved lime as possible. The time temperature of the metal before the addition of sinter should be sufficiently high.

Thus, the refining of heats by means of sinter is quite feasible. The sinter does not make the dephosphorization and desulfurization of the metal difficult, does not reduce the average rate of carbon loss in burning and does not prolong the working period. Owing, however, to some difficulties in controlling the process, the use of sinter during the working period may be recommended in exceptional cases (for instance, if iron ore is not available).

The experimental heats differed from the ordinary ones mainly in the fact that the iron ore which was to be introduced during the charging period was replaced entirely by sinter. The procedure in the deoxidation of the experimental and the ordinary heats was practically the same and, therefore, when the economics of the application of sinter were evaluated, only the difference in the consumption of limestone, lime bauxite, iron ore and sinter was taken into account.

The comparative data on the average amount and cost of materials introduced during the charging and working periods per heat show that in the production of rimmed, rail, and alloy steels with the use of sinter during the charging period, the cost of all the materials added was considerably higher than for heats in which iron ore was used. For 3kp steel, this increase constituted 3.08-6.81 rubles per ton, for rail steel – 3.14-8.46 rubles per ton, and for alloy steel – 5.0-5.63 rubles per ton. The increase in the cost of each heat took place mainly because of the high cost of sinter. The increase in the steel output owing to the reduction in the duration of the heat, and savings on permanent expenses make it possible to reduce this extra cost to 2.5-3 rubles per ton of steel. Owing, however, to the high cost of sinter, the cost of steel production is nevertheless higher.

On the basis of the work carried out, the following conclusions may be drawn:

1. The introduction of sinter instead of iron ore during the charging period does not cause any difficulties in the process of steelmaking.

2. The production of steel with the use of sinter does not affect the quality of steel adversely. When sinter was used, the average yield of first grade R-50 rails was 94.16%, and when iron ore was used it was 93.63%.

3. Sinter may be used during the charging period in the production of steel of any grade and at any furnace, either alone or in combination with iron ore. Under the conditions which obtain at the KMK, the best substitute for iron ore with respect to the iron oxide content as well as the physical properties, is the Abagursk sinter.

4. Owing to the somewhat lower oxidizing power of sinter, its consumption during the charging period is higher by 7-10% as compared with the consumption of iron ore.

5. When sinter is used in the charge, the melting of the charge proceeds steadily without vigorous reactions; during the melting period mobile slags are obtained, the amount of primary slag run off increases by 5-10%, the dephosphorization process of the steel improves, the input of the limestone during the charging period and of the lime and bauxite during the working period decreases, the duration of the melting period for 3kp steel decreases by 20 min on the average, and for rail steel by 35 min.

6. When changing to the production of open-hearth sinter, mining establishments must turn their attention especially to a reduction in the weight of sinter of per cubic meter, an improvement in its mechanical properties and reduction in its cost.

MOISTURE CONTENT OF THE GASES IN THE WORKING SPACE OF THE OPEN-HEARTH FURNACE AND THE HYDROGEN CONTENT IN THE STEEL

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Water vapor in the atmosphere of the working space of the open-hearth furnace is one of the main sources of the hydrogen which is absorbed by steel. Therefore, the dependence of the variation of the hydrogen content in the liquid steel on the actual moisture content of the gas in the working space of the furnace was studied.

The study was carried out at basic open-hearth furnaces of 25-ton capacity and 18 m^2 bottom area and fired with grade "40"-"80" fuel oil. The furnaces were provided with one double-atomizing burner on each side. The capacity of each burner was 1100 kg fuel oil per hour.

The air rate for atomizing the fuel oil was 1.0-1.2 kg/kg.

The moisture content of the atmosphere in the working space of the furnace was determined by means of a gas-intake tube (see figure), which consisted of a water-cooled 4m long tube with two coolers arranged consecutively, and a condensing copper funnel at the end. The first cooler was 3 m long and intended for cooling the gas, which was sucked out from the working space, to a temperature not below 70-80°C. After passing through the outer jacket of the cooler, the cooling water gets heated up. In the internal tube of the cooler the temperature of the water at the outlet is 70-80°C. The furnace gas, cooled down to this temperature, is not saturated with the water vapor and no condensation of moisture takes place in the first cooler. The second cooler is mounted at 45° to the first one, so that the moisture which condenses in the second cooler flows freely down into the condensing funnel. The second cooler cools the gas to a temperature of 25-35°C.

The gas from the furnaces was sucked out by means of an ejector which was connected to the gas-intake tube and provided a vacuum of 5-10 mm Hg. The volume of the gas taken was measured by a GKF-6 flowmeter.

The temperatures of the cooling water at the outlet from the first gas cooler, in the condensing funnel and in the gas flowmeter were measured by means of mercury thermometers. The condensed moisture was collected in a graduated glass cylinder which was attached to the

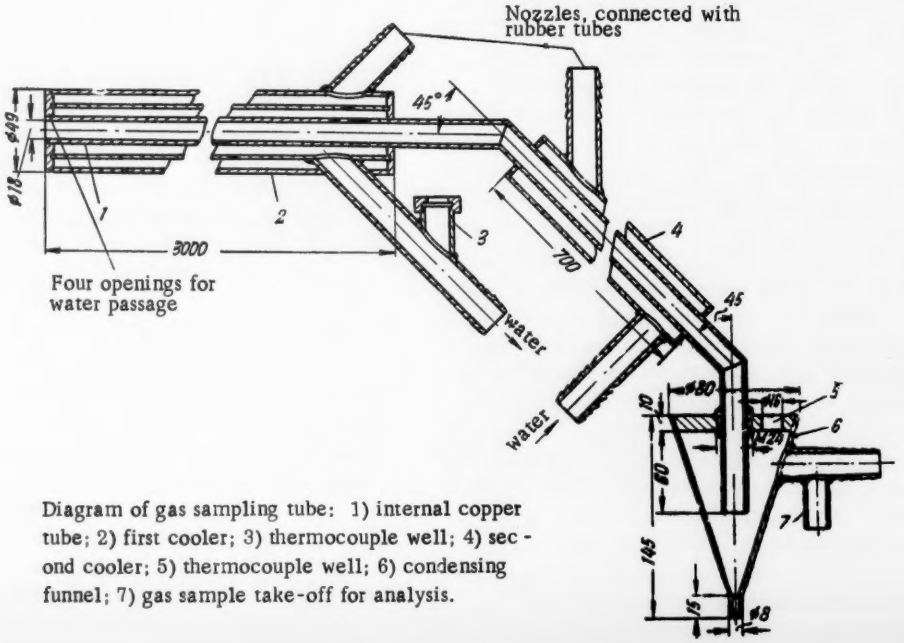

Diagram of gas sampling tube: 1) internal copper tube; 2) first cooler; 3) thermocouple well; 4) second cooler; 5) thermocouple well; 6) condensing funnel; 7) gas sample take-off for analysis.

TABLE 1. Moisture Content in Furnace Gas at the Beginning of the Pure Boil Period, %

Location of the sampling point in relation to the flame height	Heat No. 38120				Heat No. 38118				Heat No. 28107			
	base of flame	middle of flame	end of flame	entry to port	base of flame	middle of flame	end of flame	entry to port	base of flame	middle of flame	entry to port	
Above flame.	4.8	6.2	–	–	–	–	–	–	–	5.7	–	
Top of flame	8.1	8.3	10.3	11.4	7.3	–	–	–	–	7.9	–	
Center "	9.3	10.8	9.8	10.2	10.5	9.6	10.3	9.0	10.0	10.5	9.7	
Bottom "	–	–	–	10.9	11.1	10.6	11.3	–	–	–	–	

lower end of the condensing funnel in front of the gas intake of the furnace. The pressure in the system was determined by a water manometer mounted on a counter.

During the determination of the moisture content of the atmosphere of the working space, the gas-intake tube was inserted, through the peep hole in the charging door, into the furnace to a depth of 1.8-2.0 m. The cooling water in the system was at 4-5 atmos pressure. Depending on the temperature of the cooling water leaving the first cooler, the rate of gas intake from the furnace was fixed within the limits of 1-2 liter/sec. The determination was started when all the temperatures to be measured became steady and condensate drops appeared in the measuring cylinder. The time of the determination was 5-8 minutes.

Results on moisture determination. The atmosphere in the working space of the open-hearth furnace is not uniform. It is divided horizontally, as it were, into two zones. The lower zone constitutes a luminous flame spreading over the bath. The upper zone above the flame constitutes a mixture of air and combustion products. Determinations showed that the content of water vapor in the flame varies greatly over the height of the flame and increases toward the bottom. These variations are especially noticeable in the base and the middle of the flame and are less noticeable at its end (Table 1).

Initially, it was intended to measure the moisture content of the furnace gas at the very surface of the slag, in view of the fact that the hydrogen content of the liquid metal should be affected by the gas layer next to the slag surface. It was, however, not possible to carry out these measurements because the intake tube frequently became splashed with slag. In all subsequent measurements, the gas samples for moisture-content determination were taken at a height of 250-400 mm above the surface of the slag, i.e. approximately in the central part of the flame.

The variations in the gas moisture (9-11%) over the length of the flame (Table 1,2) are less pronounced than over the height.

With the object of establishing the variations in the moisture content of the flame above the bath in the course of the heat, determinations of the moisture content were made for several heats during periods which differed from each other in the fuel rate, the amount of additions, the intensity of the boil, etc. It is seen from Table 3 that the content of water vapor in the flame does not significantly vary from one heat period to another or from one heat to another.

TABLE 2. Moisture Content in the Furnace Gas Over the Length of the Flame for Various Heat Periods

Heat number	Time of sampling	Moisture content, %		
		base of flame	middle of flame	end of flame
2843	Slag making	8.8	10.0	9.0
	Middle of pure boil	7.6	8.8	9.6
3850	Slag making	9.7	11.0	9.8
	Middle of pure boil	10.3	10.4	10.0
2857	End of melt-down	–	9.7	9.5
	Slag making	–	11.2	9.6
	Middle of pure boil	9.0	9.1	9.1
38104	" " "	9.9	8.8	8.1

During the experiments, the hydrogen content of the oil was within the range of 10.8-11.5%. The moisture content of the oil fluctuated between 1 and 6%; the moisture content of the air in the atmosphere varied between 0.1-0.5% (the moisture content of the air from the regenerators was not determined). The excess air for the heats was 5-20% and for various periods of one heat the variation range was even closer.

Calculations on the fuel combustion indicate that the moisture content of the combustion products depends mainly on the hydrogen content of the fuel oil. An increase by 1% in the hydrogen content of the fuel oil, in the moisture content of the air for combustion and in the moisture content of the fuel oil raises the mean moisture content of the combustion products by 0.85%, 0.85%, 0.10%, respectively. An increase in the excess air by 10% lowers the moisture content of the combustion products by 0.8-0.9%. Since the absolute values of the factors under study vary within a relatively narrow range in one heat, the moisture content of the gas in the working space also changes very little. Direct measurements showed that the moisture content of the gas at the slag surface varies, as a rule, within the limits of 9-11%.

The mean moisture content of the gas in the working space may be different for different heats. This difference

TABLE 3. Moisture Content in the Furnace Atmosphere for Various Heat Periods, %

Period	Heat number										
	47621	37564	47643	3816	3819	2828	2833	2843	3850	2857	38104
Melt-down period	—	11.4	9.4	11.0	8.7	—	—	9.2	—	10.6	9.4
Slag-making period . . .	—	9.8	—	8.1	9.2	7.9	9.4	10.0	11.0	11.0	8.5
Pure-boil period	10.9	11.3	10.1	—	8.5	7.4	9.6	8.8	10.4	9.1	8.8
Deoxidation	9.2	10.8	9.8	—	8.8	9.8	11.2	9.7	8.3	10.0	8.0

however, very rarely exceeds two per cent. For all the heats investigated, the average moisture content of the gas atmosphere above the slag surface (Table 4) constituted 10% when the fuel oil was atomized with air, and 17% when the fuel oil was atomized with steam (steam input was 0.4-0.6 kg/kg of fuel).

TABLE 4. Moisture Content in Furnace Gas Depending on the Method of Atomizing the Fuel Oil

Location of gas sampling point	Moisture content, %				
	steam-atomized fuel			air-atomized fuel	
	base of flame	middle of flame	end of flame	top of flame	bottom of flame
Top of flame	10.8	10.7	—	—	—
Bottom of flame . .	17.3	17.4	16.4	10.4	8.8

Hydrogen content of the metal. Simultaneously with the determination of the moisture content of the gas atmosphere in the open-hearth furnace, the hydrogen content in the metal was determined in the course of the same heats. While the moisture content in the gas above the slag furnace fluctuated within the range of 9-11%, the hydrogen content in the metal increased regularly from 2-4 cm^3 / 100 g after the melt-down of the charge to 6-10.5 cm^3/100 g before the steel tapping.

On comparing the results obtained, we found no relation whatsoever between the moisture content of the gas in the furnace and the hydrogen content in the steel. Even when the fuel oil was atomized with steam, the hydrogen content of the liquid steel did not noticeably increase. It does not follow, however, that the hydrogen content of the steel is independent of the moisture content of the furnace atmosphere. The above facts indicate that the gas-liquid steel system is not in an equilibrium state with regard to hydrogen in the course of the heat but only approaches the equilibrium prior to steel tapping, and that the hydrogen content in the liquid steel is not determined merely by the partial pressure of the water vapor but depends on several variable factors. If this system were in a state of equilibrium, the hydrogen content of the liquid steel would constitute 8-11 cm^3/100 g when the fuel oil was atomized with air. At the same time, the variation of the moisture content in the gas from 9 to 11% would have an insignificant effect on the variation of the hydrogen content in the metal – not more than 1 cm^3/100 g. However, several technological factors, such as the chemical and physical properties of the steel and slag, the removal of the hydrogen with CO bubbles during the boiling, etc., hinder the establishment of the equilibrium state in the open-hearth furnace. Depending on the extent to which these factors are effective, the rate at which the gas-metal system approaches the equilibrium state with respect to hydrogen varies.

The effect of rapid variations in the moisture content of the gas atmosphere, as for instance in the case of the use of steam for fuel atomization, was not studied in the present investigation. It may be assumed that such a change in the moisture content will cause an increase in the hydrogen content of steel. But here, too, the combined effect of all the technological factors will be of decisive importance.

On the basis of the investigation carried out, the following conclusions can be drawn:

1. The water-vapor content of the gas in the working space of the furnace remains practically constant when the moisture content of the fuel varies from 1-6%, as well as when normal quantities of materials are charged into the furnace.

2. The gas atmosphere in the working space of the open hearth furnace is the main source of the hydrogen absorbed by the steel, but the hydrogen content of the liquid metal depends on the combined effect of the gas phase and technological factors of the heat.

INCREASING THE CHARGE OF THE BESSEMER CONVERTERS AT THE "KRIVOROZHSTAL'" WORKS

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Since December, 1957, the Bessemer shop at the "Krivorozhstal'" Factory has produced steel in basic converters by the method of blowing oxygen downwards onto the open-hearth pig iron. The optimum specific volume of the converter determined by Soviet as well as foreign metallurgists, for blowing oxygen downwards onto the open-hearth pig iron is 1.0-1.1 m^3/ton of pig iron, and increases as the lining of the converter gets eroded. Thus, the converters at the "Krivorozhstal'" Factory are not utilized to their full capacity and, therefore, there is considerable potential output increase.

In this connection, a number of heats of a larger weight were made (the steel from these heats was poured into five molds and not into four as was the case for the ordinary heats). The heats were carried out in accordance with the procedure accepted at the shop. The molten pig iron was poured into the converter, the first portion of the lime and ore was charged and then the blowing was started. For the experimental, just as for the ordinary heats, the oxygen input constituted 120 standard m^3/min at an operating pressure of 8-9 atmos. In all the heats the intermediate slag was run off 6-8 minutes after the beginning of the blowing. After the slag was run off, the remaining quantity of lime and ore was added and the blowing was maintained until the desired carbon content was obtained. The steel was poured through the taphole into the ladle and then it was deoxidized.

When the amount of hot metal is increased the depth of the bath increased by approximately 150 mm. For the "Krivorozhstal'" converters, which have a low (1.6) ratio of the height to the internal diameter, and which are operated as a shallow bath, this is a favorable factor with respect to the service life of the bottom and the course of the process. Oxygen was fed into the bath through a circular copper nozzle, 65 mm in diameter, with internal quadruple thread. Such a design of the lance nozzle made according to the project of the TsNIIChM personnel, ensures an earlier slag formation and, by imparting a certain swirl to the oxygen stream leaving the nozzle, helps to reduce the amount of metal particles thrown out of the converter.

An analysis of the results obtained from the experimental large-weight heats and a comparison of these results with the data on ordinary-production heats show that an increase in the weight of the converter charge is an effective way of increasing the output without impairing other characteristics of the process. Thus, the duration of the process of the large-weight heat constitutes 30 minutes on the average, i.e. it is 3 minutes or 11% longer than for an ordinary heat. When the idling periods (15%) are taken into account, the hourly output of the converter under these conditions is 10.4% higher. If the oxygen input per minute were correspondingly increased, i.e., to 140-150 standard m^3/min, the duration of the blowing period for the large-weight heat could be the same as for an ordinary heat. In that case, the hourly output of the converter would increase by 18.4%.

As is seen from Table 1, the long duration of the heat is mainly the result of the extended period of blowing. The duration of such operations as closing and opening the taphole, adding the loose materials, taking samples, and running off the final slag, remains practically unchanged, and for operations such as running off the intermediate slag and tapping the steel, it increases insignificantly. It is seen from the results given in the Table that the increase in the charge of the converter by more than 20% does not entail a proportional increase in the duration of the blowing period which increases by 12.5% only. This is due mainly to the fact that an increase in the weight of the charge favors a reduction in the relative heat losses in the converters; at the same time, it becomes possible to add a somewhat larger amount of iron ore for cooling the bath, an additional amount of oxygen being thus introduced.

A comparison of the results (Table 2) of the refining process of experimental heats and ordinary heats which were carried out at the same time, shows that in the former case the desulfurization and the dephosphorization processes not only do not deteriorate but proceed even more efficiently. A sulfur content up to 0.40% was found in 80% of the large-weight heats, whereas it was found in only 68% of the ordinary heats.

In the "Krivorozhstal'" converters, a low degree of desulfurization of steel is noted. This is explained, first of all, by a poor quality lime, which contains a considerable amount (40-50% or more) of incompletely roasted materials and sulfur (0.4-0.6%). The incompletely roasted material retards the slag formation and the removal of harmful impurities from the metal; and the high sulfur content, when the total quantity of lime added amounts to 8-9% by weight of pir iron, increases the total content of sulfur in the slag-metal system by 0.04-0.05%, i.e., the sulfur content is practically doubled as compared with its initial content in the pig iron.

It was established that, with a higher rate of oxygen absorption by the slag, the activity of the slag is increased

and the processes for the removal of harmful impurities from the bath are speeded up. The absorption of oxygen increases with increase in thickness of the slag layer. As a result, under roughly the same conditions but in narrower converters at the Petrovskii Factory where the

TABLE 1. Duration of Separate Operations in the Converter Process

Operation	Duration, min-sec	
	ordinary	exptl
Plugging the tap hole after the previous heat	1-00	1-00
Charging the hot metal	1-00	1-15
Adding the first batch of loose materials	0-30	0-30
First blowing period	7-00	7-00
Running off the slag	2-00	2-15
Adding the the second batch of loose materials	0-30	0-30
Second blowing period	9-20	11-20
Making metal and slag samples	1-30	1-30
Opening the taphole	1-00	1-00
Tapping the steel	2-30	3-00
Tapping the slag	0-40	0-40
Total duration of the heat	27-00	30-00
Duration of the blowing operation included in the total duration of the heat	16-20	18-20

surface area of the baths per ton of pig iron constitutes 0.19 m^2 as compared with 0.23 m^2 at the "Krivorozhstal'" Factory, the desulfurization and dephosphorization is somewhat more extensive. The experiments, carried out at the Petrovskii Factory, with a thicker lining, i.e., with a still smaller specific surface of the bath, gave even more convincing results, in particular with respect to the desulfurization. With the increase in the charge of the "Krivorozhstal'" converters, the specific surface of the bath decreased and the thickness of the slag layer increased correspondingly, and this affected the course of the process: the desulfurization and dephosphorization improved only slightly, though quite noticeably.

The increase in the charge did not result in a decrease in the output of acceptable product in spite of the reduction in the specific volume of the converter; on the contrary, while for ordinary heats the output of acceptable product based on the pig-iron input constituted 88% (without taking into account the iron in the ore), for the large-weight heats, carried out under the same conditions, the output of useful product increased to 89.5% (results from 106 heats). This fact was promoted

TABLE 2. Results on the Desulfurization and Dephosphorization of Heats of Different Weight

Heat	Number of heats	Mean sulfur content		Extent of desulfurization, %	Mean phosphorus content in steel, % *
		in pig iron	in steel		
Experimental	155	0.0368	0.0352	4.3	0.0254
Ordinary	168	0.0388	0.0384	1.2	0.0274

* The phosphorus content in the steel varied within a narrow range and constituted 0.09-0.11%.

by some increase in the amount of ore added, by the reduction in the metal carry-out with the thicker slag layer and by a decrease in the relative losses of metal during the slag run-off and the steel pouring.

These results once more confirm the expediency of operating the "Krivorozhstal'" converters with large-weight charge.

On the basis of the experimental heats carried out, we may draw the following conclusions:

1. The increase in the weight of the charge in the "Krivorozhstal'" converters by 20% makes it possible to increase the output of the plant by 10-18%.

2. When the large-weight heat is blown through, the purification processes of the bath from harmful impurities are improved and the output of acceptable product is increased.

3. Converters with a small height-to-internal diameter ratio (less than 2.0) and with shallow baths are less suitable for the process of blowing oxygen onto hot metal from above, than converters which are of a more oblong shape. Under the conditions which obtain at the "Krivorozhstal'" Factory, where the ratio H:D = 1.6, the use of the large-weight charge will be more economical.

THE MECHANIZATION OF LABOR-CONSUMING OPERATIONS AT THE MAGNITORSK METALLURGICAL COMBINE

A. M. Emel'yanova and V . P. Emel'yanov

Magnitogorsk Metallurgical Combine

An improvement in taking samples of open-hearth slags. In the course of the open-hearth heat, samples of slag are taken from the bath to determine the composition and the properties of the slag. These samples get very frequently mixed up. On the suggestion of V. E. Gulenkin, in order to prevent mixing up these samples, at the open-hearth shops of the combine a 20-25 mm wide and 75-80 mm long iron-plate bracket (Fig. 1) is put in the small ladle into which the slag is

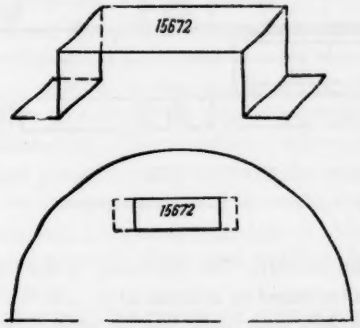

Fig. 1. Bracket with the number of the furnace and heat.

poured. The number of the furnace and of the heat is stamped out beforehand on the bracket. The lower ends of the bracket are bent out so that it does not pull free from the slag cake. The upper part of the bracket on which the number of the heat is stamped is not covered with slag. On the adoption of this measure, the check on the samples improved.

An improvement in the cooling conditions of molds. After the ingots are stripped the molds are usually cooled down to the required temperature by industrial water sprayed under pressure from holes in horizontal pipes. The existing system of cooling has several disadvantages. The concentrated impingement of the water jet, 4 mm in diameter, onto the surface of the molds creates a sharp boundary between the cooled and uncooled areas. Nonuniform temperature stresses appear on that wall and result in the appearance of cracks; transverse cracks if side-spraying is employed, longitudinal cracks if the mold is sprayed from above. The appearance of cracks decreases the service life of the molds and, hence, the molds become prematurely useless.

The holes in the pipes frequently become clogged so that the efficiency of cooling is affected and constant cleaning is required. The water does not pass through the holes which are not properly cleared and, hence, there are surface areas on the mold which are not uniformly sprayed and, as a result, cracks appear.

To bring about a uniform spraying of the molds, special sprayers were installed on the side and top spraying pipes in the open-hearth shops of the combine : two side sprayers and two top sprayers (Fig.2) were set up. The sprayers work on the principle of imparting a swirl to the water in a spiral groove. The water comes out from the sprayer in a fanlike dispersion and covers the whole surface of the mold; the whole surface of the mold is uniformly cooled. Owing to the large passages (10 mm in diameter), no clogging of the sprayers takes place.

The sprayer is of a simple design and can easily be made either by casting or forging from ordinary steel. The adoption of these sprayers increased the service life of molds by approximately 6% and resulted in a significant saving.

Stamping the mold hoops on the press. Until now, the hoops which were used for strengthening the molds when they were cast, were made from 22 mm thick steel strip by means of bending two halves when hot and then welding them. This method is very laborious and provides for a very low output.

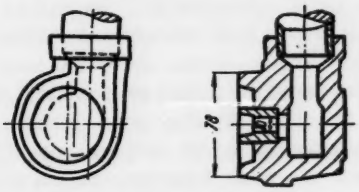

Fig. 2. Nozzle for spraying the mold uniformly with water.

A new method of manufacturing hoops on the press was mastered at the combine. For this purpose, a die and a punch (lower and upper dies) of appropriate dimensions (Fig. 3,4) were prepared. The die is welded from the plate according to the shape of one-half of the hoop. The base of the die is fastened with bolts to the block of the press. The working surfaces of the die, in the places where the strip is to be bent, are lined with "sormite" alloy to a thickness of 5 mm.

When the hoop is to be made, metal strip of the required length is placed into the die between the

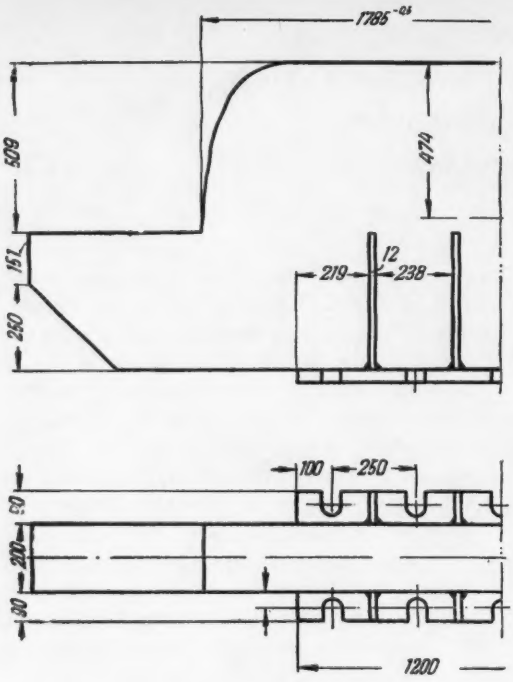

Fig. 3. Punch (upper die).

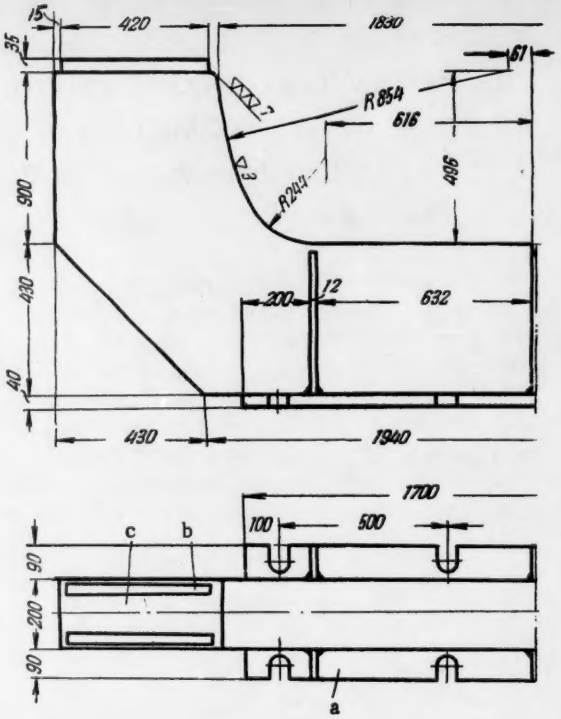

Fig. 4. Die: a) base; b) c) guides.

guides "b" and "c" and is pressed against the shaping surface of the die by means of the punch. The punch (of welded construction too) is mounted in the tool holder of the press.

The method of stamping the hoops on the press speeded up the production process for hoops and simplified it considerably. The efficiency of the bending operation increased by a factor of 6-7; at the same time, several workers were made free for other duties. This method of stamping hoops for molds may be recommended for adoption at other factories.

NEW TECHNOLOGICAL PROCESSES IN STEEL ROLLING

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The main production method for increasing the productivity of rolling mills as well as improving their mechanization and automation is to increase the extent of the continuity of all or most of the important operations in the rolling process.

Towards this aim it is planned to adopt in the near future the technique of welding together the ends of the rolled pieces to increase their length, in order to reduce the time of intervals or to achieve a fully continuous, so-called "endless" process.

As is known, this process was realized a long time ago in the USSR in the production of tubes as well as in continuous pickling, tinning and other operations; in this way the productivity of cold rolling mills is increased considerably. The adoption of butt-welding for joining rolled pieces not only increases the output of the mills, but also provides steady operating conditions for the equipment and makes it possible to solve many problems of automation in the steel rolling industry relatively easily.

The adoption of the continuous process is very economic for the hot-rolling operations. For one of the continuous mills in the USSR, a special flying welding machine was designed which moves simultaneously with the rolled piece in the course of welding. Mastering the process will provide the necessary practical experience for the extension of this method of rolling to other continuous mills in the USSR. One may expect that the new process will make it possible to increase the speed of rolling very significantly.

In the design of section mills which are to be supplied with billets from the continuous casting machine, it is proposed to apply a technological process of rolling (which is nes in principle) based on the universal planetary mill (design by A. I. Tselikov and V. V. Nosal' *, Fig. 2), with which it is possible to roll a square billet in one pass with the draft of 20-80 in one stand. Such a mill, consisting of three stands – feeding, main, and sizing – is capable of replacing about twelve stands of a continuous billet mill.

In the next five or seven years, it is planned, in the USSR, to solve the problem of combining the rolling mill with the continuous casting machine. For this end, rolling mills are being designed which will receive liquid steel through a "crystallizer", placed next to the mill, where the steel will solidify and then it will be given the required shape – plate, wire, or section steel – between the rolls of the rolling mill. The product will be delivered from the mill rolls continuously and will be cut into the required lengths in the production line.

In connection with this new feature of the future rolling mills, they should be placed near the steel melting units, and, later on, when the problem of continuous steel melting and steel pouring has been solved, they will be integrated with those units into one automatic process line.

Based on the adoption of this method, a wire drawing mill was designed in the USSR; its speed of rolling is 60 m/sec, i.e. 2-2.5 times as high as any existing mill in the USSR or abroad.

In view of the practical difficulties in welding the moving piece, especially with a high entry velocity of the initial billet to the continuous mill, it is more economic to use an appropriate loop of the metal instead of the flying welding machine if a small cross-sectional area is to be welded. At present, this type of mill is being developed in the USSR, first of all for reduction mills in continuous tube-rolling plants (Fig.1). The adoption of the butt-welding of initial tubes will permit a steady process of reducing and stretching the tube and will make it possible to produce tubes with a constant wall thickness over their whole length.

In the USSR, extensive work on the large scale adoption of the continuous casting of steel is being successfully carried out. The realization of this process will make it possible to eliminate, from the metallurgical cycle of steel treatment, several complex and labor-consuming operations on casting the ingots and rolling them on reducing rolling mills (blooming and slabbing mills). Thereby, however, the rolling operations at billet mills are not eliminated, since, with the decreased cross-sectional area of the billet cast (below 200 × 200 mm), the throughput falls very considerably. In addition the adoption of cast square (200 mm) billets at medium-section, small section and wire mills makes the installation and the operation of the mills themselves very difficult.

In that case, it would be necessary to place a large number of additional reducing stands, which would constitute a kind of a billet mill included in the train of the section mill, in front of each section mill. Such a system cannot be regarded as economic for the following reasons:

1) as regards its output, the billet mill is in a position to supply several section mills with billets, but in the aforementioned case it would be utilized only with-

in the limits of the throughput of one section mill and this would result in an increase in the weight of the equipment per ton of product;

2) the speed of rolling in those additional stands would be very low, because the piece rolled in them has a cross-sectional area. This would result not only in an uneconomic utilization of the equipment but also in an excessive cooling of steel.

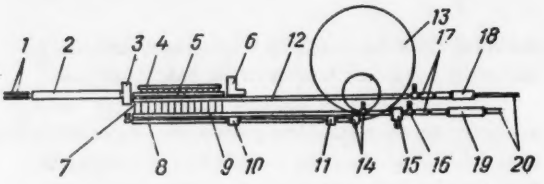

Fig. 1. Diagram of the layout of a reducing mill at which the butt-welding of tube ends is employed: 1) feeding conveyor; 2) collecting conveyor; 3) hot saw for cutting the rear end of the tube; 4) pockets for defective tubes; 5) intermediate conveyor 6) hot saw for cutting the front end of the tube; 7) transfers; 8) conveyor; 9) conveyor for moving tubes when rolled piece by piece; 10), 11) butt-welding machines; 12) feeding conveyor; 13) looper; 14) bending rollers; 15) straightening machine; 16) feeding rollers; 17) inductors for heating the tubes; 18) 10-stand reduction mill; 19) 18-stand reduction mill; 20) delivery conveyors.

In the design of section mills which are to be supplied with billets from the continuous-casting machine, it is proposed to apply a technological process of rolling (which is new in principle) based on the universal planetary mill (design by A. I. Tselikov and V. V. Nosal' • Fig. 2), with which it is possible to roll a square billet in one pass with the draft of 20-80 in one stand. Such a mill, consisting of three stands -- feeding, main, and sizing -- is capable of replacing about twelve stands of a continuous billet mill.

In the next five or seven years, it is planned, in the USSR, to solve the problem of combining the rolling mill with the continuous-casting machine. To this end, rolling mills are being designed which will receive liquid steel through a "crystallizer", placed next to the mill, where the steel will solidify and then it will be given the required shape -- plate, wire, or section steel -- between the rolls of the rolling mill. The product will be delivered from the mill rolls continuously and will be cut into the required lengths in the production line.

In connection with this new feature of the future rolling mills, they should be placed near the steel melting units, and, later on, when the problem of continuous steel melting and steel pouring has been solved, they will be integrated with those units into one automatic process line

Increasing the speed of rolling, drawing, and other production processes of rolled steel and tubes constitutes one of the most important problems in a further increase in the productivity of rolling mills. Experimental investigations showed that, with a multiple increase in the speed of steel deformation in a hot condition, the pressure on the equipment increases very little, and for cold deformation it remains almost unchanged. Therefore, an increase in the speed of rolling and drawing does not necessitate the increase of the strength of the mills and, consequently, in the weight of the basic equipment.

The following increased speeds of rolling at rolling mills, which are being designed in the USSR, are planned, m/sec:

Continuous and semicontinuous hot rolling sheet mills	15-20
Continuous and semicontinuous section mills	15-20
Continuous wire mills, ... up to	30-45
Continuous tinplate mills, ... up to	40-50
Continuous tube mills, ... up to	15

To make the continuous high-speed rolling processes possible, automatically controlled systems, by means of which a preset thickness of the rolling piece is automatically maintained, are being developed in the USSR.

For several rolling mills, the speed of rolling is at present limited by the speed of operation of the flying shears. Several types of high-speed flying shears (planetary and crankshaft-type) which are suitable for high-speed rolling operations, have been produced and are being designed in the USSR.

Increasing the weight of ingots and the draft. In the USSR, side by side with the increase in the speed of rolling and an extensive adoption of continuous production lines in the rolled steel production, there is a tendency toward increasing the weight and dimensions of initial ingots and billets.

At modern cogging (blooming) mills, the weight of ingots goes up to 10-12 tons instead of 5-6 tons, and at the slabbing mills it goes up to 20-25 tons. At light-section and wire-rod drawing mills, the weight of billets is also being increased to 250-300 kg at present.

In the USSR, it is planned to roll blooms of 400 × × 400 mm or larger cross section and slabs 2000 × 250 mm; thus a reduction in the number of passes and, hence, in the duration of the rolling process will be possible. Large blooms are rolled on large-capacity billet mills or plate mills, or on a second auxiliary blooming-mill stand which reduces the blooms without preliminary heating.

In the USSR, the duration of the rolling cycle is decreased by decreasing the intervals between passes, adopting high-speed mechanisms and improving the scheme of electric drive. Thus, the time taken to set up the upper roll on the blooming-slabbing mill has been brought down to 250-280 mm/sec. A great number of auxiliary blooming-mill mechanisms (screw-down, roller tables, manipulators, etc.) are driven according to the scheme: generator-motor.

At cogging mills in the USSR, individual drive for each working roll as well as motors with double armature are extensively used, and this substantially reduces the flywheel moment of the drive and, hence, decreases the time of reversing to 1-1.5 sec. For this purpose, magnetic, instead of electric, boosters are used in the electric drive circuit.

In order to reduce the number of passes, it is necessary to increase the draft in each pass. In the USSR, this is attained by increasing the total power of the main drive at cogging mills to 12,000 hp. The nominal torque at modern blooming mills reaches up to 180 ton-m, and at slabbing mills up to 220 ton-m.

In the next few years, blooming-slabbing mills with 1250-1300 mm diameter rolls, in which it will be possible to roll large-weight ingots at high drafts, will be built in the USSR.

When the cogging mill is rolling only slabs for the production of plates, it is most economic to install universal slabbing mills which have a higher output than the blooming-slabbing mills. For rolling slabs up to 1500 mm in width, the latter should have the top roll lifted up to 1675-1700 mm, and this operation, in spite of the high speed of the screw-down mechanism, requires considerable time during the edging passes which are necessary for reducing the side edges of the slab. The possibility of rolling billets up to 2000 mm wide at a modern slabbing mill frequently permits the omission of the spreading pass at the continuous plate mill, so that coils up to 12-15 tons in weight can be obtained, since the length of the slab is then not limited by the roll barrel length of the spreading stand. Again, the increased weight of the hot-rolled coils makes it possible to have a large-weight coil for cold rolling, up to 30-50 tons or more, with a minimum number of junctions.

As an example, let us point out that the new slabbing mill, designed in the USSR, is intended for rolling 25-ton ingots to 12-ton slabs. The annual output of the slabbing mill is 4 million tons, the weight of the mechanical equipment about 7400 tons; the horizontal rolls are driven separately by two motors of 6000 kw each; the vertical rolls are also driven by two motors of 2000 kw each; the vertical rolls are connected to the motors by means of universal couplings.

The slabbing-mill is equiped with a machine for flame scarfing all four sides of the slab.

The mechanization and automation of auxiliary operations in theproduction of rolled steel and tubes.

Whereas in the USSR and abroad definite successes have been achieved in the mechanization and automation of the rolling process, the mechanization of auxiliary operations on the inspection and finishing of the rolled product is still not adequate. These operations occupy, on the average about 65-80% of the workers engaged in the rolling mill.

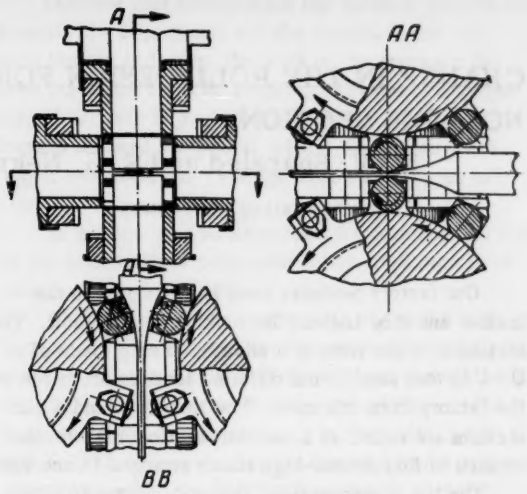

Fig. 2. Diagram of the working stand of the universal planetary mill of TsKB MM design.

A great number of original machines for auxiliary operations in the rolling process have been developed in the USSR – marking and stamping machines, machines for the delivery of sheets one by one from a pile and subsequently for stacking them into piles, separating machines which operate on an original principle and can separate sheets which were rolled in one pack, machines for binding the piles with the wire, equipment for cutting, transporting and stacking the product into piles.

Very advanced and highly productive units for the protective coating of rolled product, in particular machines for the electrolytic tinning of tinplate, for galvanizing steel sheets, for galvanizing and painting tubes, etc., are being developed and built in the USSR.

Efficient and reliable instruments for the determination of the dimensions of rolled sections and for the corresponding adjustment of the rolls of the rolling mill are being developed. The inclusion of such machines and instruments in the rolling mill and the adoption of computers for the automatic control of the mill provides the basis for building fully automated rolling mills in the USSR within the next few years.

A concentration of all the production operations will make it possible, within the next 10-15 years, to transform the rolling mill into a complex system of machines which will include not only machines for carrying out the actual process of rolling the steel and for the auxiliary operations, but also machines to carry out additional operations connected with processing the rolled product into the final article, for instance, car bodies, food cans, railroad-car axles, etc.

CHANGE IN THE ROLL DESIGN FOR ROLLING THE NO. 4 DIE SECTION

M. V. Shuraled and S. G. Nekrasov

Zlatoust Metallurgical Factory

Our factory produces steel sections used in the leather and shoe industry for making cutting tools. The section is in the form of a chamfered strip made of U7-U10 tool steel. Four different sections are made at the factory from this steel. The Nos. 2, 3, and 4 die sections are rolled on a medium-section mill which consists of four double-high stands arranged in one line.

The No. 4 section (Fig. 1) is rolled from 60 by 60 mm billet weighing 30 kg. The billet is heated in a continuous furnace fired with fuel oil; the duration of heating is one hour, the temperature 1120-1150.

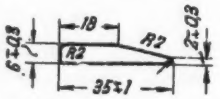

Fig. 1. Cross section of the strip.

With the old pass arrangement the section was rolled in twelve passes. It was formed from a 25.5 mm square piece in one box pass and four closed shaped passes (Fig. 2,a). This pass arrangement had the following disadvantages:

a) In the first closed pass there was a nonuniform deformation of the piece over its width: the vertical draft of the tip of the piece ($\frac{H}{h}$ = 2.32) was almost twice as big as the vertical draft of the base ($\frac{H}{h}$ = 1.21) and this caused the piece to tilt, bend or even twist in the pass; to eliminate these undesirable effects, long guides of a special form were used and the mill was accurately adjusted and controlled;

b) when the piece was rolled in the closed passes, an overfilling of the passes on the side of the base frequently occurred and sometimes caused the formation of laps;

c) it was necessary to have special (separate) rolls for the die sections in the second and third stands and, therefore, much time was lost on the roll change whenever there was a switch from the production of die section to the production of ordinary section (round, square, rectangular), and, hence, a larger stock of rolls had to be kept.

These shortcomings of the old pass arrangement made the adjustment of the mill very difficult and caused a fall in the output when the die section was rolled.

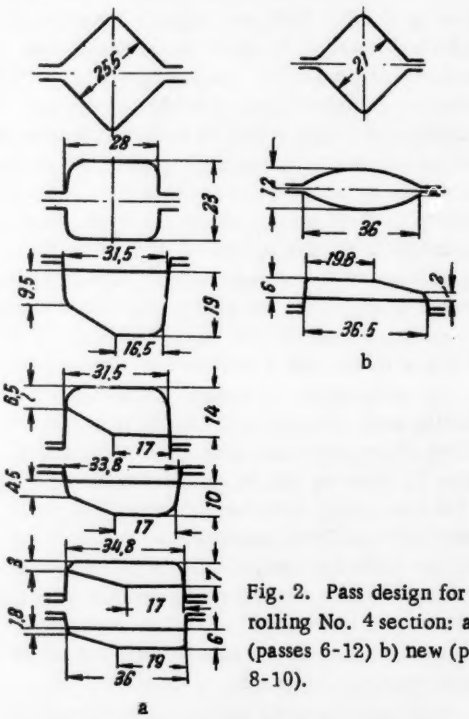

Fig. 2. Pass design for rolling No. 4 section: a) old (passes 6-12) b) new (passes 8-10).

Draft Coefficients of the No. 4 Die Section

Pass design	Draft coefficients for the passes											
	1	2	3	4	5	6	7	8	9	10	11	12
Old	1.27	1.22	1.29	1.27	1.40	1.36	1.08	1.26	1.35	1.36	1.39	1.19
New	1.27	1.22	1.29	1.27	1.38	1.32	1.36	1.29	1.66	1.49	—	—

Now, a new pass design with the use of open passes of a simple shape (Fig. 2,b) has been developed and adopted. In this pass arrangement, the section is shaped from a 21 mm square piece in the ordinary oval pass and the finishing shaped pass. When the piece is rolled in the oval groove it is displaced toward one side (in the direction of the tip of the section) so that one-sided oval section is obtained. A nearly uniform draft over the width of the piece is obtained in the finishing pass.

To reduce the probability of the formation of fins on the rolled piece, the finishing pass in the new arrangement is made slightly wider. The adoption of open grooves of a simple shape have made it possible to eliminate two passes and to increase the draft in the last passes (see table). According to the new design, 60 mm square billet is rolled into the final section in ten passes. The output of the mill has increased by 13-15%.

The new pass arrangement has made it possible to simplify the adjustment and the control of the mill considerably and, thus, the work of the operators has been made easier. The adoption of passes of a simple shape has made it possible to reduce the roll stock since there is no need for special rolls in the second stand (upper and lower pair) and the third stand (upper roll pair).

In the new pass arrangements the formation of fins in the intermediate passes and in the finishing closed pass is almost impossible.

Observations on the service life of the finishing pass have shown that its wear has increased only slightly in comparison with the wear in the old pass arrangement, in spite of the higher draft and some increase in the nonuniformity of deformation over the width of the piece. This is explained by an increase in the temperature of the rolled steel in the last pass by 40-50°C.

THE PRODUCTION OF THICK-WALLED TUBE BILLETS ON TUBE-ROLLING MILLS

A. B. Ivanov

Chelyabinsk Tube-Rolling Factory

For making hollow machine parts, couplings for drilling rods, etc. rolled steel pieces of a solid cross section are used, and, hence, in the process of manufacturing finished parts a considerable amount of steel (up to 20%) is lost as a result of machining. This can be avoided if, instead of solid bars, one uses pierced thick-walled billets made at tube-rolling mills.

The production of hot-rolled tubes with extrathick walls on existing tube-rolling equipments presents certain difficulties. The orders for thick-walled tubes accepted by the tube works do not always conform to the specifications of GOST 301-50 for general-purpose tubes with regard to the upper wall-thickness limits.

The main causes which limit the possibilities of tube-rolling plants with regard to the production of thick-walled tubes are: technically possible minimum deformation during the piercing of the tube billets in the rotary-piercing mills, the limited strength and durability of the mandrel-support bars, inadequate capacity of the reducing and sizing mills, and some other causes.

Until recently, a draft coefficient of less than 1.4 on the rotary-piercing mill was considered impossible. However, experiments carried out at the Nikopol Novotrubnoi Factory, Pervoural'sk Novotrubnoi Factory and the Baku Factory showed that under suitable conditions one can attain smaller drafts and thicker walls.

As early as 1951, during experimental rolling operations at the Nikopol Works, a 246 × 70.5 mm shell was obtained by piercing 250 mm round billet of the 45 steel, and that corresponds to a draft of approximately 1.26.

In 1957, in connection with the assignment for the production of 146 × 40, 108 × 28, 89 × 24, and 76 × 19 mm tubes, the Baku Tube-Rolling Factory carried out experiments on the determination of the possibility of producing extrathick-walled tubes on automatic tube-rolling machines.

To carry out the piercing operation involving a minimum draft, the design of the working rolls of the piercing mill was modified (Tables 1 and 2). As compared with the ordinary rolling process, the parameters of the mill adjustment were also changed.

For the production of thick-walled tubes with small diameters, initial billets of an appropriately smaller diameter are used. However, the smaller the diameter of the billet, the more difficult it is to carry out the piercing operations with a minimum draft.

The inadequate drawing forces at the contact surface of the rolls necessitate an increase in the draft, and that leads to an increase in the elongation and, consequently, to a reduction in the wall thickness. The modified design of the rolls made it possible to carry out the piercing process with drafts which do not exceed 8-10%. Such a rolling operation, with the above roll design showed a number of advantages as compared with the old roll design. It became possible to omit the extra change of rolls when changing from the production of ordinary to thick-walled tubes.

TABLE 1. Design of the Working Rolls

Templet designation	Characteristics					
	R	III	R_1	α^0	r	Mean diameter
RT-1	62.25	129.0	106.0	25	6	127
RT-2	60.50	125.5	103.0	25	6	123
RT-3	58.75	122.0	92.0	25	6	119.7
RT-4	57.25	119.0	91.0	25	6	116.7
RT-5	55.75	115.5	98.0	20	6	113.5
RT-6	54.25	112.5	110.0	20	6	110.5
RT-7	53.00	109.8	112 3	20	6	107.9
RT-8	51.75	107.0	111.0	20	6	105.2
RT-9	50.50	104.0	108.0	20	6	102.5
RT-10	49.25	101.0	97.0	20	6	99.7
RT-11	48.00	98.5	95.5	20	6	97.2
RT-12	47.00	96.0	86.0	20	6	95.0
RT-13	46.00	93.5	80.5	20	6	92.7
RT-14	45.25	91.5	61.0	20	5	91.0
RT-15	45.00	90.0	—	—	5	90.2
RT-16	49.25	98.5	96.7	20	6	97.7
RT-17	48.50	97.0	—	—	5	97.0
RT-18	52.00	104.8	110.0	20	6	104.4
RT-19	51.60	103.2	109.0	—	5	103.2
RT-20	54.75	110.5	111.0	20	6	110.0
RT-21	54.75	109.5	—	—	5	109.5
RT-22	57.75	116.5	92.0	20	6	116.0
RT-23	57.75	115.5	—	—	5	115.5

TABLE 2. Arrangement of Rolls According to Templet

Diameter of finished tube	Stand														
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
89	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
96	1	2	3	4	5	6	7	8	9	10	16	17	—	—	—
102	1	2	3	4	5	6	7	8	18	19	—	—	—	—	—
108	1	2	3	4	5	6	20	21	—	—	—	—	—	—	—
114	1	2	3	4	22	23	—	—	—	—	—	—	—	—	—

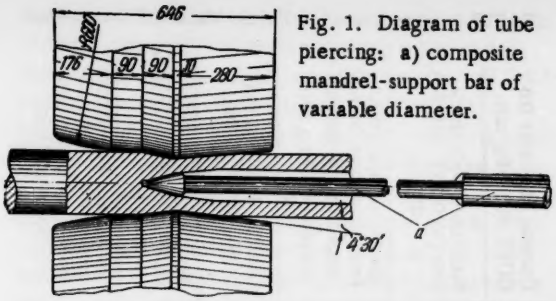

Fig. 1. Diagram of tube piercing: a) composite mandrel-support bar of variable diameter.

When the initial billet of a smaller diameter is used one must, even if taking into account a subsequent size reduction, decrease the diameter of the mandrel accordingly and, consequently, the diameter of the mandrel-support bar. It is, however, not possible to use long, thin bars in view of their inadequate strength to take a considerable axial load during piercing. To make the bars more durable and stronger they were made composite from two or three parts of different diameter with a smooth, tapering transition from one part to another (Fig. 1). The lengths of the thinnest part of the bar was equal to the lengths of the thick-walled shell delivered from the mill.

The results of the experimental piercing operation (Fig. 2) made it possible to conclude that the elongation of 1.26 during the piercing in the rotary-piercing mill is still not the minimum one, and under certain conditions, it could be made still smaller.

For smoothing out the spiral marks on the outer and inner surface of the shell and for obtaining a smooth surface on the tube when the ordinary method of tube production is used, the shell is subjected to longitudinal rolling on the automatic mill and to transverse rolling on the reeling machine before being sized or reduced. The wall of the tube is thereby reduced by 2-3 mm. If the shell is not subjected to longitudinal rolling on the automatic mill and, that operation being omitted, the tube is subjected only to transverse rolling in the reeling machine with subsequent sizing or diameter reduction, then a gain in the wall thickness (up to 3 mm) will be achieved; at the same time, the wall thickness does not exceed the tolerance limit even when the piercing marks remain on the tube surface.

An increase in the wall thickness by 10-15% before the sizing or reduction mill may be obtained by a preliminary reduction with the use of small-diameter mandrels on the automatic and the reeling machine; this method is sometimes used by tube manufacturers when a thick-walled tube is made from a large diameter billet on the sizing or reducing mill.

For the prevention of variation in the wall thickness during the production of thick-walled tube special attention was paid to the positioning of the mandrel at the center of metal deformation in the piercing mill. At the moment when the billet comes into contact with the mandrel it should be in the exact center position in the axis of piercing. For lifting the mandrel rod, at the moment when the piercing operation is commenced, to the level of the axis of piercing, the pulling front roller was placed slightly higher than the ordinary roller. To prevent the mandrel from deflecting down, an extended mandrel-bar support was used. From this point of view, the use of a permanent, water-cooled mandrel, which has been adopted at several tube works, is very desirable.

The reducing mill plays a special part in the production of thick-walled tubes. Since the reduction of the tube can be effected on 20 stands of a powerful reducing mill, it is possible to produce thick-walled tubes from initial billet of a diameter which exceeds the diameter of the finished tube by 45-50 mm.

However, the generally accepted distribution of drafts for normal-wall tubes was found unsuitable for the reduction of extrathick-walled tubes; it was necessary to change the design of the reducing-mill rolls, involving considerably smaller increases in drafts (with regard to the height and the width of the pass) from one stand to another.

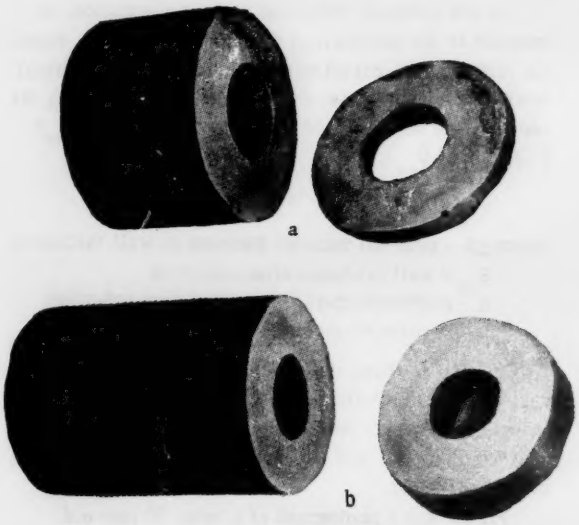

Fig. 2. Cross section of the shell after piercing: a) 178 × 51 mm shell obtained from 180 mm diameter billet with 66 mm diameter mandrel; draft = 1.248; b) 168 × 49 mm shell obtained from 170 mm diameter billet with 61 mm diameter mandrel; draft = 1.24.

The design of the rolls of the UZTM reducing mill for rolling thick-walled tubes of 89-114 mm diameter from billets of 120-140 mm diameter is given in Fig. 3 and Tables 1 and 2. During the introduction of this pass design, in accordance with the method described above, 89 × 27 mm tubes were rolled. The initial material was 120 mm round billet, the mandrel of the piercing mill was 63 mm in diameter, the shell on the piercing mill was 122 × 25 mm in size, the pass of the automatic mill measured 116 mm, and its mandrel 67 mm.

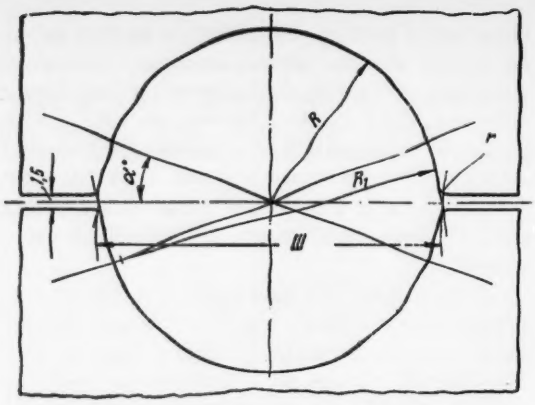

Fig. 3. Roll design of reducing mills.

After the rolling on the reeling machine, the reduction to the final dimension of 89 × 27 mm was carried out on 10 stands.

If, instead of 120 mm diameter billets, one uses the billet of 140 mm diameter, it is possible to obtain tubes 89-114 mm in diameter and 33-35 mm in wall thickness from 125 mm pass of the automatic mill.

In the course of reduction without elongation, an increase in the thickness of the wall takes place. From the results of the experiments carried out, an empirical relation for the increase in the wall thickness during the reduction of thick-walled tubes was obtained for $S_{tu} = (0.15 - 0.5) D_{tu}$:

$$\Delta S = 0.014 S_{tu} \cdot n,$$

where ΔS - over-all increase increase in wall thickness;
S_{tu} - wall thickness after reduction
n - number of operating stands during reduction
D_{tu} - outer diameter of tube

The tube rolling shops of the Pervoural'sk , Zakavkaz, and Baku factories, which have more powerful reducing mills, can produce thick-walled tubes of 48-121 mm diameter and wall thickness equal to 0.30-0.35 outside diameter.

Figure 4 is a photograph of a tube, 70 mm o.d. and 27 mm wall thickness (i.e., wall thickness equal to 0.385 o.d.), obtained by means of reduction rolling. The tube was rolled from 120 mm diameter billet in the 116 mm diameter pass of the automatic mill with a subsequent reduction in 15 stands of the reducing mill with ordinary-design rolls.

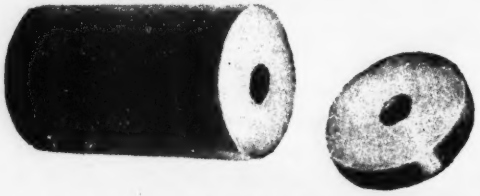

Fig. 4. 70 o.d. tube obtained by means of reduction rolling.

TABLE 3. Assortment of Thick-Walled Tubes Rolled

Outside diameter (to GOST 301-50), mm	Maximum wall thickness (to GOST 301-50), mm	Maximum wall thickness which can be obtained on the automatic tube-rolling mill, mm	Outside diameter (to GOST 301-50), mm	Maximum wall thickness (to GOST 301-50), mm	Maximum wall thickness which can be obtained on the automatic tube mill, mm
48	—	17	121	28	36
57	13	19	127	30	38
60	14	20	133	32	40
63.5	14	21	140	36	42
68	16	23	146	36	44
70	16	24	152	36	45
73	19	25	159	39	48
76	19	26	168	45	50
83	19	28	180	45	54
89	24	30	194	45	58
95	24	31	219	45	60
102	24	32	245	45	65
108	28	34	273	45	65
114	28	35	—	—	—

At present, the Baku Tube-Rolling Factory produces 73 × 19, 98 × 24, and 108 × 28 mm tubes which have been ordered.

The thick-walled tubes of 48 and 57 mm diameter may be produced by combining the sizing of the tubes in the sizing mills to 83 mm diameter and a subsequent reduction to the final cross section; at the same time, tubes with a wall thickness of up to 0.35 outside diameter can be obtained.

For instance, a 48 ×17 mm tube may be rolled from a 105 mm diameter billet in the 94 mm pass of the automatic mill and, subsequently sized to 83 mm in 5 stands of the sizing mill, and reduced in 20 stands of the reducing mill. This method will make it possible to establish a continuous tube-production line, but it requires an additional means of transport for returning the tubes from the sizing mill to the receiving rack or to the trough of the reheating furnace of the reducing mill.

The presence of several modifications in the method of producing tubes with a large variations in the tube-wall thickness at a plant containing sizing and reducing mills requires a clear planning of operations and special care in the adjustment of the equipment in order to minimize the production time losses.

Medium and large tube-rolling plants have no powerful reducing mills and, therefore, the production of tubes of 127 mm, or larger outside diameter, with a maximum wall thickness is possible if a minimum draft is obtained in the piercing mill while the initial billet of a maximum possible diameter is used.

For instance, on one of the mills at the Baku Factory it is possible to roll tubes of 127 mm diameter with 33 mm wall thickness from billets of 160 mm diameter,

and tubes of up to 40 mm wall thickness from billets of 170 mm diameter. At the tube-rolling mills of the Nikopol and the Zakavkaz factories, one can roll thick-walled tube shells, conforming to GOST 301-50 specifications, of 168-273 mm o.d. and wall thickness equal to 0.25-0.30 of the outside diameter of the tube. The possibilities of these mills with regard to the production of thick-walled tubes of outside diameter above 273 mm are limited by the minimum drafts as well as by the maximum diameter (300 mm) of the initial billets which can be obtained at the tube-billet mill.

The assortment of thick-walled tubes which can be made at tube-rolling plants with automatic mills is given in Table 3.

Editor's note. The results of the experiments carried out at the Nikopol, Pervoural'sk, Novotrubnoi, and Baku factories, on the production of thick-walled tube shells at tube-rolling mills, require further confirmation in commercial-scale production.

THE MECHANIZATION OF LABOR-CONSUMING OPERATIONS AT THE MAGNITOGORSK METALLURGICAL COMBINE

V. P. Emel'yanov

Magnitogorsk Metallurgical Combine

Water-cooled bar for cleaning the furnace bottom. Up till now the bottom of the reheating furnaces in the thin-sheet mill have been cleaned with bars of circular solid cross section. The built-up deposits of scale were eased off with the bar and knocked off the bottom by means of a hammer. The bar used to become hot and tended to bend, so that the furnace operators frequently sustained burn injuries to their hands.

A special hollow, water-cooled bar (Fig.1) was adopted at the Combine on the suggestion of V. N.

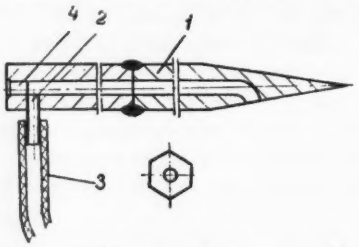

Fig. 1. Water-cooled bar: 1) old drills; 2) branch tubing; 3) hose; 4) bar base.

Gur'yanov. For this purpose, old hexagonal-section drilling tools with holes were used. The bar consists of two drills welded together. The opening at the end which is hit by the hammer is closed by means of electric welding, and a branch tubing is fitted at the side. When the water hose is attached to the branch tubing, water is let through the bar. At the other end of the bar, 40 mm from the sharp edge, an opening is drilled through to serve as a water outlet.

The water-cooled bar does not become overheated and does not bend; thus, the working conditions for the furnace operators are improved and saving in metal can be effected. In addition, water which passes through the bar to the sediment on the furnace bottom promotes the establishment of a high difference in temperature between the bottom and the scale sediment, thus causing high internal stresses leading to cracks and lamination of the sediment. This improves the effectiveness of the bottom cleaning and increases the operating efficiency.

During furnace operation, the bottom is cleaned by means of a specially shaped steel plate (Fig. 2) which is pushed, together with the slabs, and placed above the spot where there is scale sediment. The bar is passed] under the plate and the sediment is removed by means

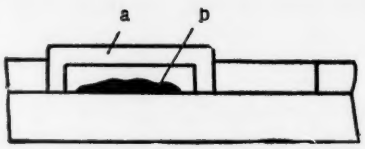

Fig. 2. Shaped plate for cleaning the furnace bottom; a) plate; b) scale deposit.

of a hammer. The plate is then pushed toward the furnace outlet and taken away.

Owing to the adoption of the shaped plate, the heat emission and flame bursts from the burners do not present substantial obstacles during bottom cleaning: the plate separates the section with the scale deposit from the hot zone of the furnace.

As the water outlet is at a certain distance from the bar tip and is on its side, there is no possibility of the outlet becoming plugged. The adoption of the cooled bar resulted in a substantial improvement in furnace bottom cleaning. Similar bars may be used at any reheating furnace.

Electromagnetic rollers in front of the shears. Light steel sections are cut, by means of shear (Fig.3), to uniform length in batches consisting of a few pieces. The shear stop is set up according to the required length of the section to be cut. After they have been cut, the pieces are conveyed on rollers to the collecting pockets.

Previously the end crops (3-4 m long) which remained in front of the shears after the cutting operation, were not taken away by the rollers so that they had to be pushed by hand, and, hence, the operating efficiency of the shears was adversely affected. In addition, when a large amount of steel sections was cut, the operator had to hold the bending and vibrating ends in front of the shears, and that could result in an injury.

On the suggestion of V. F. Shvetsov and V. M. Kruglov, two electromagnetic rollers were installed in front of these shears. The body of each roller contains electromagnetic poles (Fig.4). When electric current is sent through the coils, a magnetic field is established between the poles. The poles are separated from each other by means of laminae made of diamagnetic alloys. Electric current for the coils, wound from 3 mm diameter wire, is supplied continuously through collecting rings during the whole operation of the shears. When the shears and roller tables are stopped, the supply of electric current to the coils is cut off.

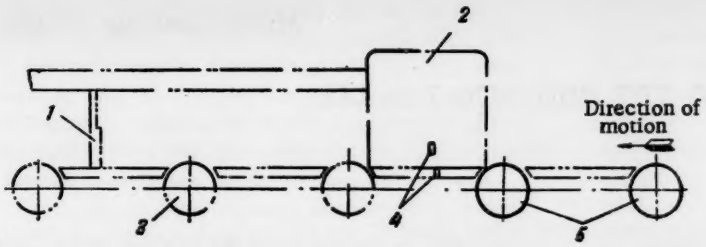

Fig. 3. Diagram of the positioning of shears and rolls: 1) stop; 2) shears; 3) rolls; 4) knives; 5) electromagnetic rolls.

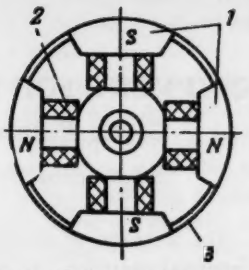

Fig. 4. Electromagnetic roll: 1) poles; 2) coil; 3) foils.

Now, there is no need to push the remaining ends after the cutting; in this way, the throughput capacity and the efficiency of the shears has been increased. The pieces, held tight against the electromagnetic rollers, do not spring up during the cutting and, thus, the work of the assistant operators has been made easier and safer.

The conveyance of the rolled pieces on the roller tables has been made more effective and it is now possible to stop the roller tables rapidly in an emergency.

EQUIPMENT FOR ENTERING THE ROD INTO THE DIE

G. M. Fomin

Magnitogorsk Sizing Factory

The technique of drawing steel rods involves, as an essential operation, the tapering of the end of the bar over the length of 200-220 mm so that it may be easily threaded through the die hole and gripped by the draw block. The tapered end of the rod, however, has to be cut off during the subsequent operations since it cannot be utilized.

The resulting metal losses depend on the length of the rod and on the length of the pointed end; they reach up to 3-4%, and, in large-scale production, amount to hundreds and thousands of tons. In addition, a large number of workers has to be engaged on the operation of tapering the ends and special equipment has to be provided.

With the view to reducing metal losses on end crops, and to reducing labor expenditure on the preparation of steel rods for drawing, work is carried out at home and abroad on the use of reliable and efficient methods of entering the end of the rod into the die without tapering the rod beforehand. Rather bulky machines, requiring special equipment and driven by a separate hydraulic drive, have been developed.

The author of this article, together with the employees at our works, Yu. T. Bulynenkov and V. Ya. Zagvozdin, designed and put into operation an equipment for entering the rods of various steel grades (10-50) of 13-35 mm diameter, by making use of the idle pass of the drawing frame (see figure).

At the moment when the rod is entered into the die, the steel is compressed at the central point of deformation. At that time, the forces applied to the rod are several times higher than during the ordinary drawing. As its length from the place of the compression to the die is relatively large (200-250 mm) and its diameter fairly small (15-23 mm), the rod end may bend and fail to enter the die. To prevent rod of these dimensions from bending, the pushing-in block is designed in such a way that the rod end which is pushed in is 150-175 mm long. When thicker rods (23 mm or more) are drawn (pushed in), this operation is possible if the end of the rod to be pushed in is 200-250 mm long. For a better grip of the rods by the jaws and for the elimination of jamming of the pushing-in jaws, the guiding grooves are made at an angle of 9 deg. Flat roller bearings are mounted between the guiding planes of the block and the jaws. The equipment ensures a reliable operation even if instead of the roller bearings, one uses two hardened plates of which one is attached to the guiding plane of the block and the other to the gripping jaw.

The new equipment has the following advantages over the old design: 1) no separate electric and hydraulic drives are required; the idle pass of the drawing

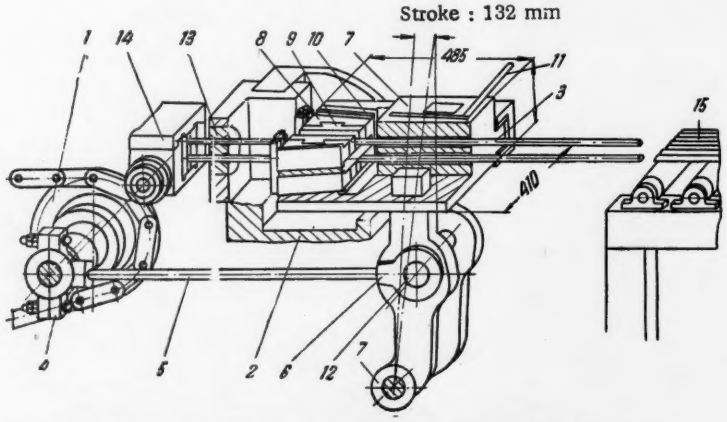

Diagram showing the method of entering the rod into the die: 1) sprocket wheel with two eccentrics; 2) welded base fixed with bolts to the front support of the drawing machine; 3) body of the pushing-in block; 4) crosshead; 5) connecting rod; 6) link; 7) push rod; 8) push jaws; 9) jaw insert; 10) roller bearing; 11) lever for tightening the jaws; 12) shaft connecting the link and the push rod; 13) die; 14) draw block; 15) rack.

machine is utilized for pushing the rod into the die; 2) the rods can be drawn in two strands with the full use of the mill capacity; 3) the design is simple and compact; 4) there are no indentations exceeding the limits of tolerances permitted by GOST; 5) production cost of the equipment is low; the equipment can be made at any factory, since it does not involve complicated parts.

At present, there are seven drawing machines (six machines of 20-ton capacity and one of 15-ton capacity), which are provided with the pushing-in equipment, operating at the Magnitogorsk Sizing Factory. In the near future, the equipment for rod entry will be installed on three more drawing machines.

By providing the chain-drawing mills with the equipment for entering the rod into the die without the need for tapering the rod end beforehand, it will be possible to reduce the steel consumption, to eliminate partly, or completely, the shops for tapering the rod ends, and to make tens of tapering lathes and hundreds of lathe operators available for other duties.

THE EFFICIENCY OF STEEL ROPES

B. M. Vorontsov

Leningrad

The most extensively used steel ropes are cross-wound ropes made of strands laid into a rope; the rope is regular lay. The pitch of the rope strands is not the same in all the concentric layers and, therefore, the wires in those rows cross over each other, having a point of contact; as a result, shearing stresses appear between the adjoining layers in the strand. This partly explains the rather low service life of the ordinary rope.

In the second method of steel rope twisting, the wires in all rows of the strand are laid in such a way as to have a line of contact. The pitch of winding in all concentric rows remains equal.

The line-contact ropes can be made only by using various-diameter wires in the strands. Each consecutive layer of the wire is laid into the grooves formed by the wires of the preceding layer (for example GOST 3077-55), or a part of the wires is laid into the grooves of the preceding layer and another part follows the crest of the inner layer (for example GOST 2688-55), in both cases having a line contact between the layers. Sometimes, use is made of both methods simultaneously (for example GOST 7668-55).

There are no shearing stresses in a line-contact rope and, hence, a higher percentage of the cross-sections area of the rope is occupied by steel; therefore, with the same diameter of the rope and the same ultimate tensile strength of the wires, line-contact ropes have higher breaking strength. Thus, rope of 22 mm diameter (GOST 2688-55) has a cross-sectional area of all the wires equal to 184.5 sq mm, and, according to GOST 3070-55, the cross-sectional area is equal to 175.56 sq mm, i.e. in the first case the cross-sectional area is higher by 5%.

The service life of ropes in which the wires are in line contact is higher than that of ordinary ropes and, therefore, it is more advantageous to use them. Thus, only line-contact ropes are used for excavators; the life of these ropes is higher than ordinary ones by a factor of 1.5-3 (depending on the operating conditions). A wear test of ropes in which the wires are in line contact (6 × 19 + 1 mm), with the use of pulleys at a ratio D:d equal to approx. 20 (where D is the diameter of the pulley and d the diameter of the rope), showed that their service life was higher than that of ordinary ones by a factor of 1.5, and in the tests in which a fixed number of the wires of the rope was broken, it was higher by a factor of 2.

The ropes in which the wires are in line contact are given the following designations:

LK – the rope in which the wires are in line contact in the strands;

LK-O – wires of one diameter in individual layers of the strand;

LK-R – wires of two different diameter in the outer layer of the strand;

LK-RO – wires of different diameters in some layers of the strand and of one diameter in other layers;

LK-Z – wires of a smaller diameter fill the spaces between two layers of larger-diameter wires.

For making a long-life rope, it is necessary to lay the wires in a stress-free condition or at least partly free from internal stresses.

There is a certain method of rope twisting in which a preliminary deformation of wires is applied and thereby their internal stresses are decreased.

The deformed wires show no tendency to reverting to the initial form and, therefore, there are no factors causing the rope to untwist into its constituting elements; such a permanently twisted rope will not untwist when its ends are not anchored, and the broken ends of the wires will not tend to stick out.

A special attachment (consisting of 3 roller sections) to the ordinary rope-laying machine is used for deforming the wires or the strands. On going around the roller, before being stranded, the wire is deformed and, when this deformation is combined with the rotation of the wire during the stranding in the plane which is perpendicular to the former one, the wire is shaped into a definite helix which corresponds to its position in the strand.

Laboratory tests on permanently twisted ropes and observations of their durability in service have shown that they are superior to ordinary ropes. There are instances when the service life of the permanently twisted rope is 5 times as high as that of ordinary ones.

Permanently twisted ropes of 39 mm diameter (GOST 7668-55), when used for large-capacity excavators, exhibit 2-3 times as long a service life as ordinary ropes.

Laboratory tests have shown that 6 × 19 + 1 mm ropes, when used for service on pulleys of ratio D:d = 20 until 10% of the wires are fractured, exceed the durability of ordinary ropes by a factor of 1.5 to 1.7.

Such a diversity of service characteristics of permanently twisted ropes can, apparently, be explained by the fact that a single standard method for their manufacture has still not been developed. The above-quoted characteristics refer to Lang-lay ropes in which the wires are in line-contact. In every case, a superiority of the deformed-wire rope over the ordinary one has been established.

THE STORAGE OF SPARE PARTS OF EQUIPMENT

I. E. Moshkevich

Dnepropetrovsk Metallurgical Institute

When the system of planned, preventive repairs is introduced at a metallurgical factory the organization of the storage of spare parts – the determination of the standard quantity of spares, the service life, transportation and the storage of parts - becomes of extreme importance. However, not enough attention is being paid to this problem even now. At several metallurgical factories (the Kirov, "Azovstal' ", Enakievo and others) no individual records of spare-part movement are kept at storage places.

The amount of spare parts and units available at the store and made in the repair shops and sections of the factory, and their consumption during repairs are usually determined in tons for various types of product and in accordance to monthly turnover reports prepared by the shop offices on the basis of the reports obtained from mechanics and repair foremen at the shops.

With this method of keeping the records, the chief mechanics' departments are hardly in a position to check if the orders placed for the parts correspond to the actual needs of a shop. This is partly the cause of an accumulation of unnecessary spare initial material, semifinished and finished parts which lie untouched in the shop and factory stores for several months (or even years) while there is a constant shortage of essential parts.

A quantity of spare parts which are kept in store for a long time are written off every year as there is no more use for them because of reconstruction, modernization of machinery, etc.. For instance, at the machine shop of the Petrovskii Works, initial material as well as semi-and fully finished parts amounting in value to 1.7 million rubles were written off as scrap in 1957.

Evidence on the use of returnable package materials, issued for handling spare parts, also indicates that the spare parts are kept in the shop stores for too long a time. For instance, the number of rounds (i.e., the number of times a given item circulated) of the standardization equipment in the main shops of metallurgical factories in the South is 1-2.2 per year.

For the purpose of limiting the tendency of the shops to increase the amount of spare parts in their stores as well as for the most economic distribution of repair services, "quotas" ("limits") of lathe-hours for machining the parts for separate shops are alloted beforehand at several factories (the Petrovskii, Kirov, Enakievo, and other factories). These quotas are alloted on the basis of statistical data on the actual services of the repair-and-machine shop in the past, on the coming major repairs of the equipment, on the planned productive capacity of the machine shops and other factors.

The shops have to make out applications and orders for spare parts and units in conformity with the allotted quotas.

There is no doubt that the planning of the work in the repair-and-machine shop on the basis of the quota system, introduced some order into the organization of spare-part preparation. Nevertheless, each shop aims at using up its allotted quota regardless of its actual needs.

The lack of an itemized record of spare parts leads to an incorrect representation of the production cost calculated according to the directive, "Current repairs and maintenance of basic equipment", and hence, frequent discrepancies occur between the actual stock of parts and the records of the account department. For instance, at the Petrovskii Factory, in the rail and structural shape mill alone, spare parts which were not used in the production processes and amounting to 800,000 rubles were revealed during inventory in October, 1957.

The situation as regards the storage of spare parts (in particular medium and large size) is no better. The organization of storage has very substantial shortcomings.

Thus, the first task for the works is to introduce some order into the record keeping and storage of spare parts, the establishment of norms for the service life of spare parts and the minumum stock. The establishment of norms is a particularly involved and difficult task depending on the systematizing of the basic data for the determination of the service-life norms for spare parts.

In order to accumulate statistical data for the determination of service-life norms for frequently replaced (so-called "running") spare parts and to ascertain the requirements in spare parts, one can determine their service life by experimental analysis. To prevent excessive safety limits in the service-life norms of spare parts, the results of the experimental analysis should be corrected on the basis of the study of information on the number of parts ordered and made during a certain period as well as on the basis of the records on the average amount of spare parts in stock and their consumption, if itemized records of the spare parts movement is kept.

It is desirable to maintain a reserve stock, first of all, of those parts whose service life is equal to or somewhat longer than the normal time required for making them at the works or obtaining them from a supplier.

In this way, one can determine the minimum quota (which should provide for the requirements in spare parts during the whole time while the new assignment of parts is manufactured) by means of multiplying the length of time (in months) required for the manufacture of the parts (including the time period for making out the order preparing the model, initial material, setting up the plan of work in the repair-and-machine shops) by the monthly requirement in parts, which requirement is taken to be the monthly consumption of these parts.

It is suggested that the following scheme should be introduced to systematize the spare-parts stock. The equipment teams of the chief mechanics' department, together with the shop heads' assistants responsible for the equipment, or the shop mechanics, should determine the list of spare parts which should be kept constantly on hand in stock. The guarantee stock in a store should be limited only to the parts whose service life is approximately equal or slightly longer than the time required for making them at the factories.

The main shops of the K. Liebknecht Factory introduced a certificate in which the characteristics of the part, a sketch, and the record of the movement is included for each part. Such a method of record keeping makes it possible:

a) to determine the technical characteristics and to obtain the data required for the establishment of the service life and the stock quota for every part in storage;

b) to determine the availability of the parts in stock and the time when orders should be made for replenishing the stock;

c) to systematize the materials for analysis and for workingout the measures designed for increasing the wear resistance and determining the cost of parts, as well as the necessary returnable materials for storing the parts;

d) to obtain data on the monthly turnover of parts; the turnover may be determined from the ratio of the cost of the parts which were issued from the store in a month to the cost of the total stock of spare parts.

From the example of leading machine-construction factories, it seems to be essential to establish a procedure according to which parts should be replaced in exchange for worn ones, in order to ensure the collection and inspection of parts (especially small-sized parts) with a view to reconditioning them by various methods (machining, polishing, milling, etc..) to new dimensions, or using the worn parts as initial material for making parts of a smaller size. Unfortunately, enough use is still not being made of worn parts at many metallurgical factories.

When group repair shops or central repair shops are set up at several factories, there will be great scope for centralizing all the preparatory work for repairs, including storage and keeping a record of the spare parts in those shops. For this purpose, however, regional mechanized stores for keeping small- and medium-size parts should be established.

The decentralization of storage and record keeping of a huge number of parts at shop stores and floors which are not properly equipped for that purpose results, at present, not only in an increase in the number of staff dealing with stores, orders, etc., but also in an increase in the reserve stock on one-type parts.

Side by side with the systematization of the spare-part stock, it is necessary to start work on the standardization and unification of parts with a view to reducing the variety of types of parts. Following the example of machine-construction factories one should, for this purpose, have in the design sections of the factory a card index of certificates of the parts made at the factories. This would enable the personnel of the design department and the OGM, when preparing drawings for reconstruction for major repairs or for the modernization of equipment, to refer to existing models (this procedure was followed at the Petrovskii Metallurgical Factory before WW II).

A RATIONAL UTILIZATION OF WELDING SLAGS

G. P. Peichev and N. S. Kormilitsyn

VNIIOChERMET

At present, in connection with the fall in the supplies of open-hearth iron ores available in the Urals and the Krivoi Rog Basin, deliveries of iron ore of class 21 for metallurgical factories have decreased. The open-hearth shops frequently have to carry out heats using ordinary ore which has a high silicon content and a large amount of fines and dust, which increase the amount of slag and contribute to a premature breakdown of the checkers.

In May, 1957, when considering the problems of economy in steel production, the All-Union Conference of Steel Melters recommended the use of more economic raw materials, fuel and other materials, as well as industrial wastes and by-products of metallurgical establishments. Scale and welding slag, in which the iron content exceeds 65-70%, constitute the waste products of rolling mills. The Vyksa Metallurgical Factory is the pioneer in the application of welding slag as an oxidizing agent during the refining of the open-hearth heat. There, use is made of welding slag which forms in the reheating furnaces of the light-section and plate mills and which is collected in special cast-iron or steel bins, as well as of welding slag which is removed during the scheduled repairs of reheating furnaces. At the open-hearth shops of the Vyksa Factory, about 170 tons of molding slag is consumed every month (2040 tons per year). The cost of one ton of prepared slag is 6 rubles. As a result of the use of welding slag instead of iron ore the factory saves about 234,000 rubles annually.

The use of welding slag is specified in the technological instructions on making St. 1, St. 3 and St. 6 grade steels and causes no difficulties in the dephosphorization and desulfurization of steel even when the sulfur content in the slag is high. A high specific weight of slag (5.3) as compared with iron ore, improves the contact of the welding slag with the metal without vigorous reactions.

At the Kuznetsk Metallurgical Combine, crushed welding slag from the soaking pits of the blooming mills was used in the finishing period of the heat, as well as in the charge together with the ore during the experimental production of killed carbon steel. At the same time, special attention was paid to maintaining a normal slag regime.

According to the data of the Central Factory Laboratory of the Kuznetsk Combine, when welding slag with a silicon content higher than in the ore is used, it is necessary to increase the input of limestone during the charging period or of lime during the finishing period of the heat. At the same time, it is pointed out that welding slag, as well as rich iron ore, increases the yield of liquid steel, as a result of a high degree of reduction to metallic iron. The additions of welding slag in the charge together with iron ore caused no difficulties or complications in furnace operation.

From the experience of these two works it is seen that the use of welding slags in open-hearth production is expedient and constitutes an additional method of reducing the cost and increasing the yield of liquid steel.

Several factories, however, do not attach serious importance to the collection, preparation, storage and utilization of welding slag.

One should bear in mind that each ton of welding slag replaces 1.3-1.2 tons of scarce open-hearth ore and this means a saving of 60-70 rubles for each ton of slag used. In this way, by using welding slags it is possible to save several million rubles in one year if one considers the present volume of rolled steel production at work in our country. Iron and steel works and National Economic Councils should pay more attention to the collection, preparation, and utilization of welding slags as a very effective means of reducing the cost and increasing the output of steel.

THE HISTORY OF FURNACE TECHNIQUE IN RUSSIA

I. B. Gol'denberg

(The work of M. Moiseev "On Industrial Furnaces", 1845).

In 1845, "The Mining Journal" (Nos. 10,11, and 12) published an article "On Industrial Furnaces " by Mikhail Moiseev, Grade II Captain of the Mining Engineers Corps. Although the author of this article referred to the lectures by Lieutenant Colonel G. A. Iossy and to the book "System der Metallurgie" by Karsten, which he has used as sources it should be stressed that Moiseev's work is an original analysis of the information collected by the author, and it considerably excels the sources quoted with regard to the depth of the analysis.

Moiseev, in the same way as his remarkable contemporary, P. P. Anosov, attached great importance to the design and operation of furnaces. The study of oper tional experience at industrial furnaces enabled Moiseev to substantiate several theses which are still valid nowadays. In particular, he gave a very accurate definition of an industrial furnace as "an apparatus consisting of refractory material and serving the purpose of treating iron ores and industrial products by means of the action of an elevated temperature brought about inside this apparatus by means of burning any combustible material".

Moiseev divides all industrial furnaces into three types: shaft furnaces, flame furnaces (natural draft or air), and vessel furnaces.

At the same time, Moiseev pointed out: "shaft as well as flame furnaces, with regard to the shape and location of parts, comprise a large variation of types,but with all this diversity there is a common similarity in the design of all shaft and flame furnaces and this similarity will be considered here in more detail".

This statement is the starting point in the general theory of furnaces. Indeed, the problems of fuel use (combustion), heat transfer, gas movement, design of standard components, utilization of the heat of flue gases, etc., are common to all furnaces. The state of science at that time did not enable Moiseev to solve the problems of combustion, heat transfer, and aerodynamics which became a part of the furnace technique as late as the twenties of this century. However, the problem of a "common similarity in the design of all shaft furnaces and flame furnaces " was posed quite correctly.

A part of Moiseev's work dealing with shaft furnaces appeared in "The Mining Journal" Nos. 10 and 11 (1845). Here, the author considered, in detail the components, size and type of furnaces,their equipment and instrumentation, and the operation of shaft smelting furnaces which include, first of all, the blast furnaces. This work enables one to form quite a clear idea of the design and operation of shaft furnaces in the beginning of the nineteenth century.

On the basis of numerous observations, Moiseev pointed out that the "...combusted material is more effective in furnaces which have thick outside walls than in furnaces with thin walls".

This is a clearly expressed statement that the thicker the brickwork of the furnace (other conditions being equal), the smaller are the conduction losses of heat through the walls. One must bear in mind that the statements, quoted above, from Moiseev's work are based on the operational experience of then-existing furnaces and refer to that period of scientific development where the so-called "phlogiston theory" was still widely accepted.

The work "On Industrial Furnaces" contains a detailed description of various forms of fuel, and well-founded reasoning for the amount of " compressed air" (air fed under pressure from the air blowers) which is essential for the operation of the blast furnace.

Apparently, many experts are unaware that more than a hundred years ago Moiseev studied the effect of the addition of steam to the blast in the operation of the shaft furnace and concluded that the action of water vapor consists in the "change of the place where it is desired to have a high temperature",. In other words, when steam is fed to the shaft furnace and there is an adequate total quantity of heat, it is possible to move the high temperature zone within the volume of the furnace.

Further on, Moiseev considered the problem of the service life of furnaces, dealt in detail with the use of mixed gases which evolve during the operation of smelting shaft furnaces and gave their approximate chemical composition (13% CO_2, 23% Co, 5% H_2 and 59% N_2). Moiseev recommended the following rule which defined the condition for the utilization for combustion of the gasses which evolve during the operation of shaft furnaces. "If the sum $CO + H_2$, expressed in %, is greater than 25, then the gas can burn".

A part of Moiseev's work deals with flame furnaces and contains a description of the component parts and design of reverberatory furnaces, problems in the erection of those furnaces, and thecommon design elements for all flame furnaces, viz., roof, fire grate, hearth, arch and chimney.

The problem of the use of fuel occupies a fair part of the book. Moiseev remarked that the amount of fuel charged into the furnace at one time is determined by the draft and other factors (furnace operation). Then he says "all other conditions being equal (and with the same design of the fire chamber), the combustion of the fuel is more efficient in those furnaces where a higher temperature is attained. Incomplete combustion is a mixed

process which consists of the combustion proper and of the carbonization or destructive distillation of some portion of the fuel".

After a short description of the operation of flame and annealing shaft furnaces, Moiseev described the design and preparation of vessel (crucible) furnaces. He devoted special attention to the problem of constructing "melting pots". In this field rich experiences was accumulated in the course of steel production by his predecessors (S. I. Badaev) and his contemporaries (P. P. Anosov and others).

Moiseev's work "On Industrial Furnaces" appeared thanks to extensive experience accumulated by the Russian metallurgical industry in the course of preceding years, including the eighteenth century when Russia was the world leader in pig-iron output. One should not fail to mention the part played by A. Ketov, L. Simbirtsev, D. Krokhalev, A. Subboten and many other still unknown Russian innovators in steelmaking and in constructing original furnaces at the Verkh-Isetsk, Nizhne-Isetsk, Elizaveto-Nordvinsk, Neva, Votkinsk, Bui and many other factories at the end of the eighteenth and the beginning of the nineteenth centuries.

Between 1840 and 1845 the "Mining Journal" published five articles by Moiseev including "The Design of English Furnaces for Steel Production" (No. 5, 1840), "On Hardening and Annealing Steel for Various Purposes" (No. 1, 1843), and "On Phenomena Developed by the Flame in Existing Puddling and Welding Furnaces" (No. 11, 1845).

A VISIT TO KOREAN METALLURGISTS

P. Druzhinin

"Elektrostal'" Factory

As a member of the Soviet trade-union delegation, I visited the Korean People's Democratic Republic from October 24 to November 13, 1958. During my stay, I visited metallurgical factories in Kanngye, Kansen, and Kim Chak.

At the metallurgical combine in Kanngye, the blast-furnaces are operated with iron-coke *. The blast-furnace shop of the combine is expanding its productive capacity all the time. A large blast furnace is now under construction. The open-hearth furnaces are being reconstructed with a view to doubling their capacity. I was told at the combine that the output of rolled steel has increased several times over as compared with the output during the Japanese occupation of Korea.

At the Kansen Metallurgical Factory a large scale construction is under way. At the end of 1958, a new electric furnace was put into operation, and in the first quarter of 1959 one more electric furnace will be started. The construction of a Bessemer converter with side blast is nearing completion. The electric blast furnace shop will be put into operation in 1962.

A Bessemer shop, a tube-rolling mill, a blooming mill, and a large oxygen plant are being built at the works. In the first quarter of 1959 a wire drawing shop is to be put into operation. The steel output of the factory in 1959 will exceed the steel output of 1958 by a factor of 3.4 (at the same time the number of employees at the factory will remain the same). For the most part, the production expansion will be achieved through an extensive application of oxygen, the mechanization of labor consuming operations, and an improvement in production methods.

The Kim Chak Factory is justly called a "quality-steel factory". Here, the production of a hundred grades of steel was mastered in a short period, and, in 1959, the personnel of the factory is to master the production of 200 more grades of steel.

The factory is fully equipped with modern steel-rolling, forging, and stamping machinery; there are good research and analytical labatories.

At present, with the participation of Soviet engineers, a plate-rolling factory is under construction. The mill for this factory was made at the Elektrostal' Heavy Machinery Factory. Eleven electric furnaces at the Kim Chak Factory are designed for charging from above.

The great successes of Korean metallurgists are due, to a great extent, to the fact that Korea now has her own qualified personnel. Under the Japanese occupation there were no qualified Korean personnel; at present, about 300 engineers and technologists who rose from the ranks are employed at the Kansen Factory alone. An evening institute is attached to the factory and about 200 workers receive training. The director of the factory, Comrade Li Dyai Chen, previously worked as a rolling-mill operator, and received practical training in the USSR at the "Elektrostal'" Factory. The chief engineer of the factory, Comrade Kon Hen Ku, was previously a steel smelter.

In Korea, Socialist competition is spreading and the advanced practices of leading workers in industry are widely disseminated.

Professional meetings were organized at the Kansen Factory and the Kim Chak Factory. I related my experience and gave our Korean friends a few hints regarding the modification of equipment and methods of steelmaking. In the course of our talks, the Korean metallurgists showed great interest in the methods of oxygen application for the intensification of metallurgical processes in the USSR.

There are still some shortcomings in the operation of Korean factories, in particular the technology of making some grades of steel is not fully worked out, there is a shortage on high-carbon scrap, and the express laboratories have not got their full complement of qualified personnel. However, all these are temporary shortcomings. The patriotic enthusiasm of Korean metallurgists, guided by the Labor Party of Korea, will assist in eliminating these shortcomings in the very near future.

* More details on the operation of blast furnaces with iron-coke in Korea will be given in Engineer Pak Syn Nok's article which will appear in a later issue of our journal. Editor.

NEW BOOKS

I. G. Grinshpunt, The Cold Drawing of Steel. Moscow, Metallurgizdat, 1958, 120 pages.

The book contains a description of the technology of steel drawing. The processes in preparing the steel for cold drawing (pointing, annealing, straightening, etc.) are discussed in detail. The operating conditions for drawing round, shaped, and other sections are explained; the assortment and tolerances of cold-drawn steel rods are given; the causes of defective products are analysed and methods of preventing and eliminating them are considered. The book contains a description of the design of the main types of equipment for tapering and drawing, and of heat-treatment shops as well as of the manufacture of drawing equipment.

The book is intended to raise the technical standard of workers at steel-drawing shops.

S. S.

A. I. Tselikov and V. V. Smirnov. Rolling Mills. Moscow, Metallurgizdat, 1958, 432 pages.

The book is a revised and supplemented edition. It contains a description of the construction and design of rolling mills for various purposes (from cogging to tube-rolling and thin-sheet rolling mills), of their parts and auxiliary machinery and mechanisms. A part of the book deals with special-purpose mills.

In the book, the lubricating systems of rolling mills, the main types of lubricants and methods of feeding the lubricant to the frictional surfaces, are considered in detail.

The book is well illustrated. To give a clearer understanding of complex mechanisms, the descriptions are supplemented with kinematic diagrams.

The book is intended as a text for metallurgical and mechanical engineering institutes and can be used as a handbook for mechanics, technologists, designers, and engineers working in industry and in design organizations.

S. S.

V. A. Kulebin. The Preparation of Ores for Smelting (2nd edition, revised and enlarged). Moscow, Metallurgizdat, 1958. 518 pages.

The first chapter of the book is devoted to the characteristics of iron, maganese, and chromium ores as well as fluxes employed in the ferrous industry. In the second chapter, the operations in preparing ore for smelting are considered: crushing, size-separation, size reduction, washing, beneficiation, roasting, sintering and briquetting. In one of the sections, the blending of raw materials is described. A separate section is devoted to the selection of sites for ore beneficiation plants and to the operational experience of the Krivoi Rog Yuzhni Ore Mining and Beneficiation Combine.

The third chapter of the book deals with auxiliary operations on the preparation of ores for smelting. It contains a description of the design of feeders, conveyors, elevators, tube conveyors, thickeners, filters, etc. Schemes for dealing with tailings at beneficiation plants, the arrangement of bins and storage are described. The last section of the chapter is devoted to control and testing procedures at ore beneficiation plants.

The book may be useful for a wide circle of readers – miners, operators of ore beneficiation plants, metallurgists and economists, as well as for students of metallurgy.

A. G.

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosénergoizdat	State Power Engr. Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LÉIIZhT	Leningrad Power Inst. of Railroad Engineering
LÉT	Leningrad Elec. Engr. School
LÉTI	Leningrad Electrotechnical Inst.
LÉTIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MÉP	Ministry of Electrotechnical Industry
MÉS	Ministry of Electrical Power Plants
MÉSÉP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTi	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NIIZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOÉ	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIÉL	Central Scientific Research Elec. Engr. Lab.
TsNIÉL-MÉS	Central Scientific Research Elec. Engr. Lab.-Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIÉSKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZÉI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us - Publisher.